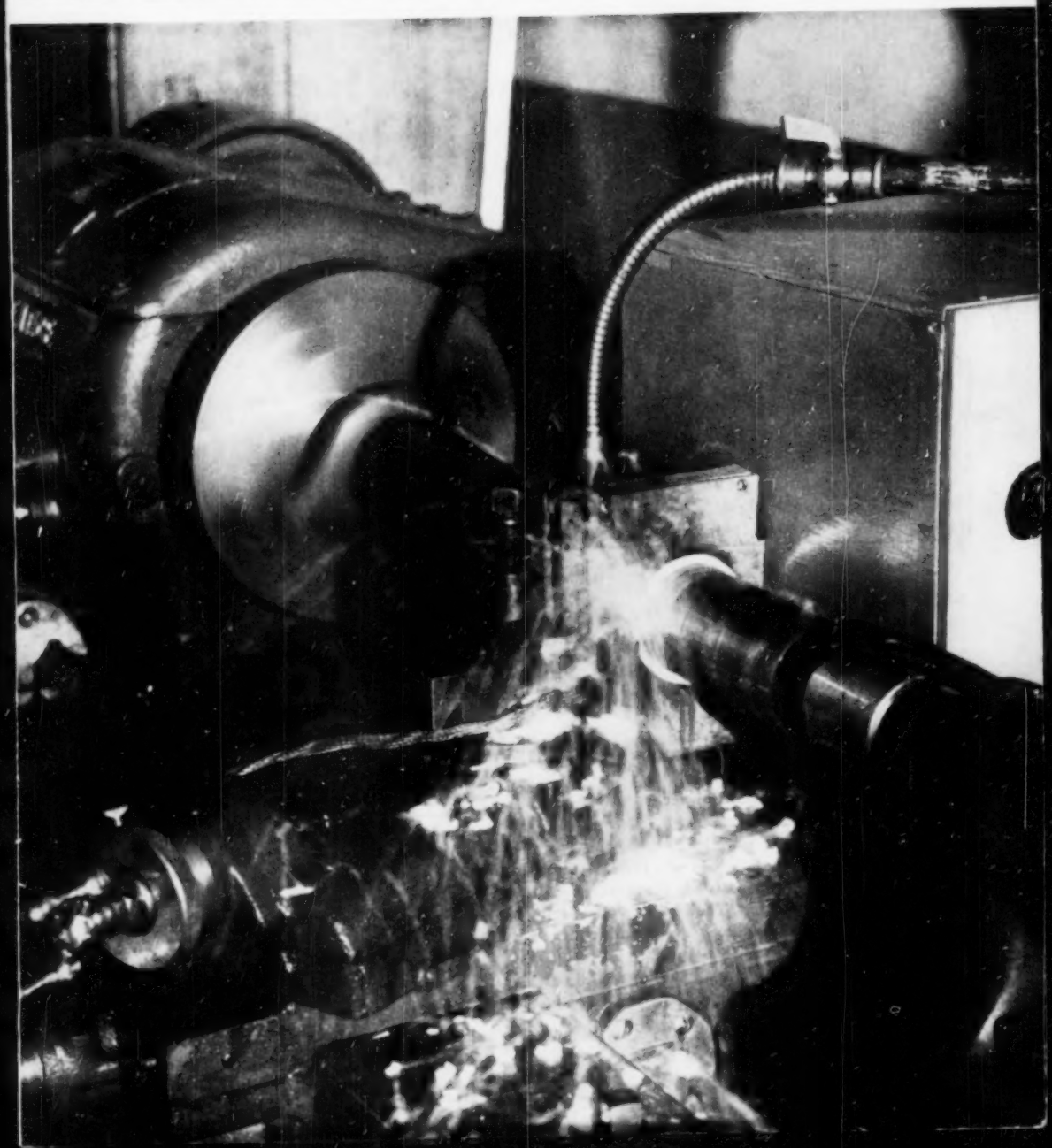
MAY 1950

Journal of METALS



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VOL. 188, NO. 5 MAY, 1950

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COVER

Machining at workpiece temperatures up to 1500°F permits faster cutting, lower power requirements, and heavier tool feeds and speeds. Metallurgical studies show that such heating has no ill effects on the hardness, surface condition, or microstructure of finished parts. S-816, a high temperature service alloy, is shown being turned at 9 cu in. per min at the Sam Tour & Co. laboratories.

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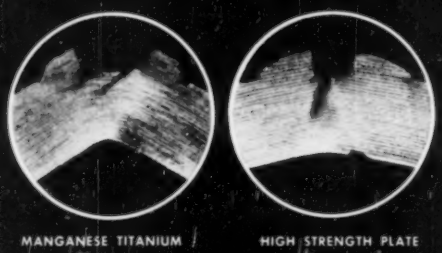
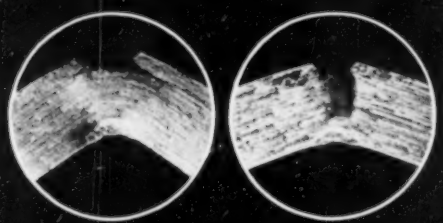


MANGANESE TITANIUM — HIGH STRENGTH PLATE

V-Notch machined prior to bending



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Extra-Low-Carbon STAINLESS STEEL

New Type Chromium-Nickel Steels Have Added Corrosion Resistance

New and improved austenitic stainless steels of the 18-8 type have been developed which have superior corrosion resistance after being exposed to heat. These steels, known as extra-low-carbon stainless steels, were designed especially for use in welded and stress-relieved equipment that is exposed to more severe corrosive conditions than are normally encountered by other types of straight 18-8 stainless steel.

Under severe corrosive conditions, intergranular attack may occur in the higher carbon unstabilized grades of austenitic stainless steels that have been subjected to the temperature range of 800 to 1600 deg. F. during welding or hot forming operations. It is generally agreed that this type of corrosion is caused by complex carbides that are formed at the grain boundaries of the stainless steel during heating.

The effect of heat is rarely harmful in the ordinary fabrication of stainless steel for most applications, such as in architecture, the food and dairy industries, in hospitals, and in the home. However, in the chemical and other allied industries, where stainless steel is used in the handling of very corrosive chemicals, these new extra-low-carbon stainless steels should most certainly find wide use.

Fig. 1 Left: Carbide precipitation at the grain boundaries of an 18-8 stainless steel, containing 0.059 per cent carbon, after being held at 1200 deg. F. for 1 hour. Right: Absence of carbide precipitation in 18-8 stainless steel of 0.03 maximum carbon content, after being held at 1200 deg. F. for the same length of time.

In general, there are three ways in which the precipitation of carbides can be controlled in stainless steel:

1. Heat-treating so that the carbides present are dissolved.
2. Alloying with an element, such as columbium or titanium, that will tie up the carbon in the form of a harmless carbide.
3. Decreasing the carbon content of the steel.

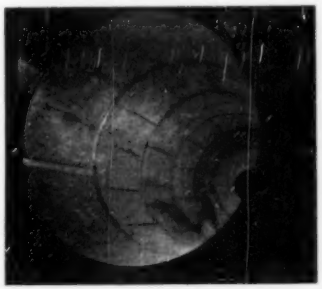

Fig. 2. The new extra-low-carbon stainless steels are especially suited for large types of process equipment, such as this fractionating tower. They require no heat-treatment after welding.

Heat-Treatment After Welding

Before the development of extra-low-carbon stainless steel, or of the "stabilized grades", one means for preventing intergranular corrosion was to heat-treat stainless steel that had been subjected to the dangerous temperature range, so that the precipitated chromium-carbides would go back into solid solution. It was found that when a welded part was heated to temperatures of 1950 to 2000 deg. F., and then cooled rapidly, most of the carbides were retained in solid solution. This extra heat-treatment is sometimes impractical, however, because of the design or massive size of some types of welded equipment.

Use of Columbium and Titanium to "Fix" Carbon

As the result of a search for a method of producing stainless steels that would be immune to intergranular corrosion without heat-treatment, the columbium- and titanium-bearing stainless steels were developed. These elements form carbides more readily than chromium. It was found that columbium, when present in a 10 to 1 ratio to carbon, completely "fixes" the carbon and renders it harmless in stainless steel. A similar effect can be accomplished by alloying with about five times as much titanium as carbon. Steels thus alloyed with columbium or titanium are known as "stabilized grades."

Decreasing Carbon Content

The most recent development in preventing intergranular corrosion has been the extra-low-carbon stainless steels. To be substantially harmless in stainless steel for as-welded or welded and stress relieved chemical equipment operating at temperatures under 700 deg. F., carbon must not be present in quantities over 0.03 per cent.

In 1937, ferrochrome with 0.03 per cent maximum carbon was first produced for the steel industry by ELECTROMET. This product has made it possible to produce very-low-carbon stainless steels—steels that are completely immune to intergranular corrosion when welded or subjected to a stress-relieving heat-treatment.

Metallurgical Service Available

If you use welded stainless steel equipment, it will pay you to investigate the advantages of using extra-low-carbon steels. If you produce stainless steel, our metallurgists will be glad to give you technical assistance in the use of ferrochrome of 0.03 per cent maximum carbon. For further information, write to the nearest ELECTROMET office.

For a more detailed account of the properties of extra-low-carbon stainless steel, write for a free copy of the technical paper, "Resistance to Sensitization of Austenitic Chromium-Nickel Steels of 0.03% Max. Carbon Content".

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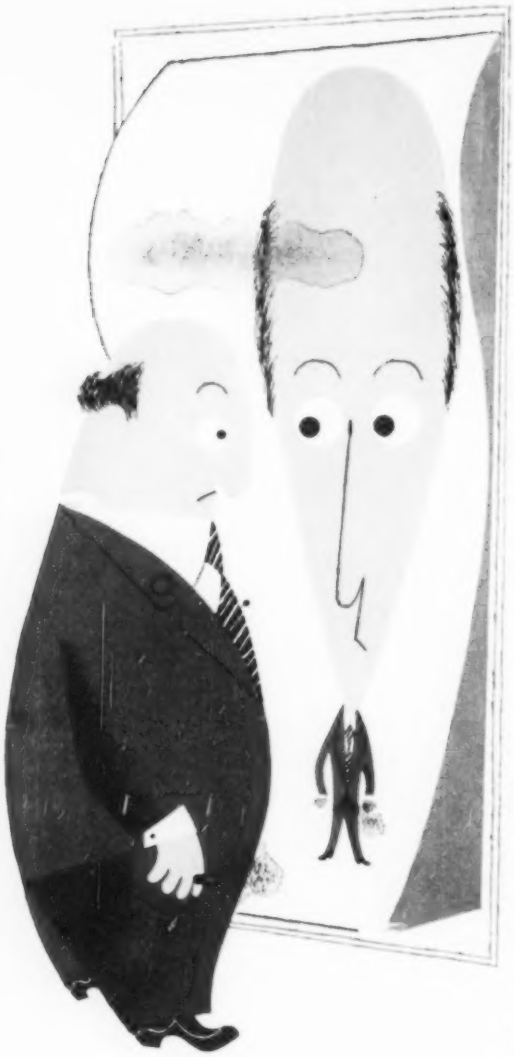

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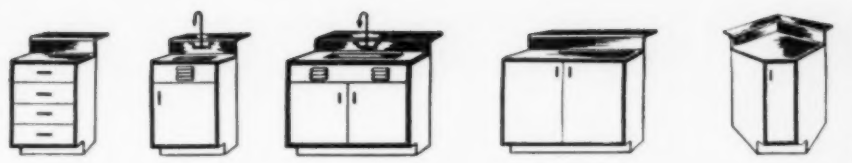

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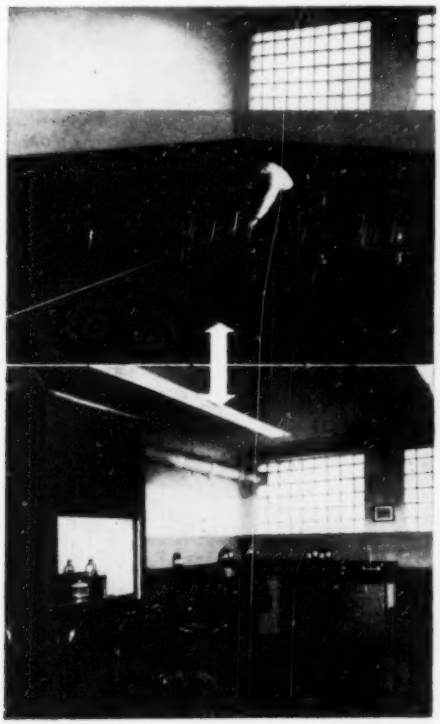

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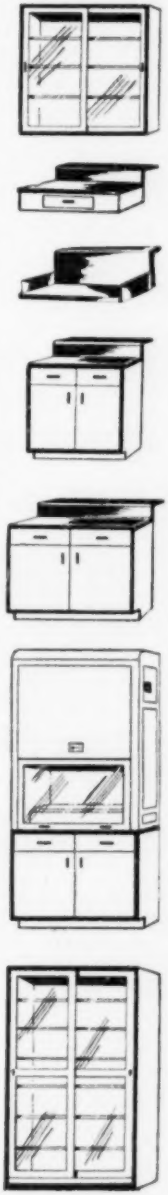

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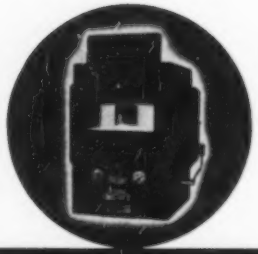

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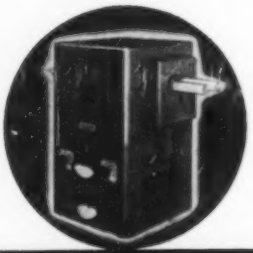

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Authors In This Issue

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W. E. Hammersla

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J. H. Calbeck

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W. T. Maidens

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(Continued on next page)

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Iron and Steel Division May Dinner

The regular annual May Dinner of the Iron and Steel Division of AIME will be held, as usual, on the night before the A.I.S.I. banquet, May 24th at 7:00 p.m. at the Engineers' Club, 32 West 40th Street, New York City. Chairman H. K. Work has obtained Cyril N. Hoyler, Research Engineer, R.C.A. Laboratories, as speaker. His subject is "Electronic Arithmetic" and his talk will include demonstrations and motion pictures of the principles and practices of high speed computing.

All AIME members and guests are welcome, at $6.00 per person, including cocktails. Return postcards are being sent to ISD members east of the Mississippi, but if you do not get a card for this stag dinner, write Ernest Kirkendall, Secretary, Iron and Steel Division, AIME, 29 W. 39th Street, N. Y. 18, N. Y., indicating the number of reservations desired.

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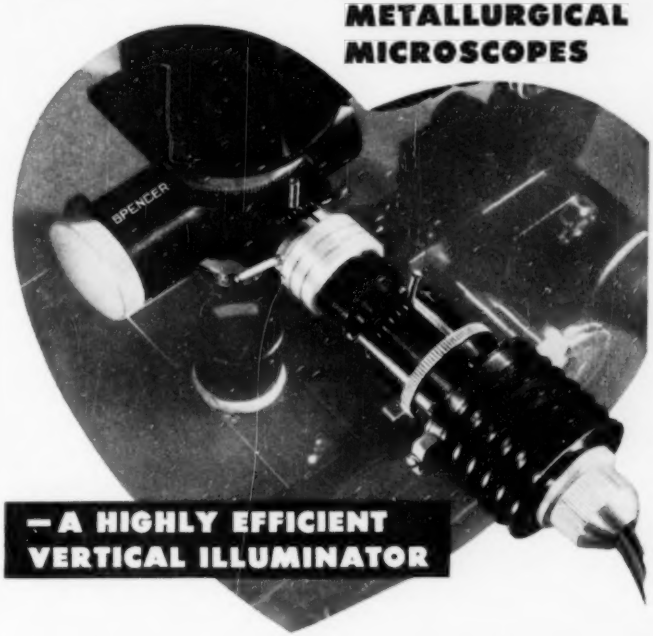

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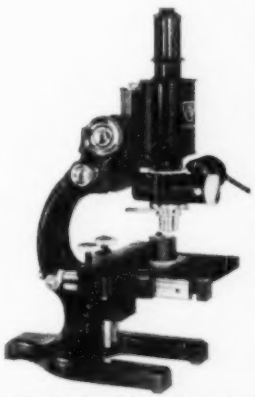

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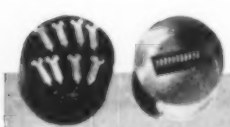

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LEWIS B. LINDEMUTH

Consulting Engineer

140 CEDAR STREET, NEW YORK 6, N. Y.

STEEL PRODUCTION

DESIGN · OPERATION · CONSTRUCTION · METALLURGY

For Sale

ELECTRIC MELTING FURNACE

Top charge, 9-ft. diam. shell, with 2500 KVA transformer. Very little used, in good condition.

FRANK B. FOSTER, INC.

2217 Oliver Bldg. Pittsburgh 22, Pa.

R. S. DEAN LABORATORIES, INC.

Consulting, Research Development
Chemistry, Electrochemistry, & Metallurgy
Laboratory Research on a Contract Basis

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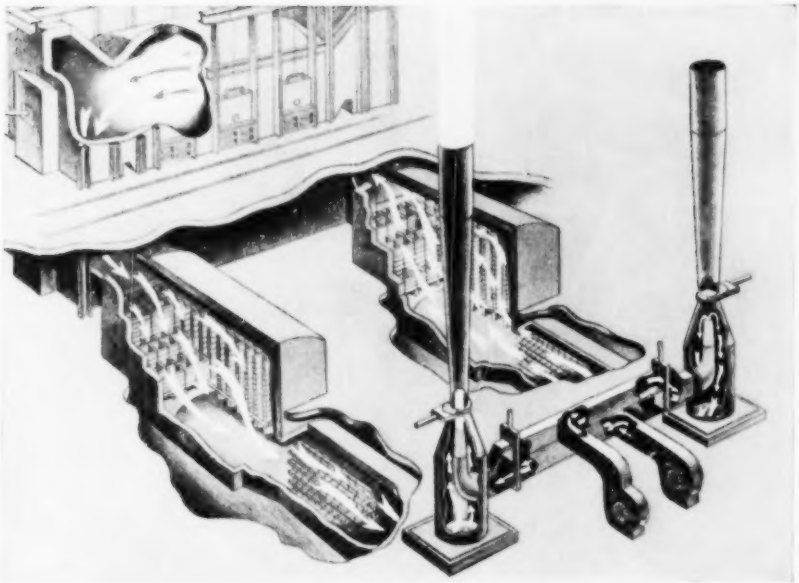
RIVERDALE, MD.



An Equation Worth Knowing:

Morgan-Isley
Furnace Control System
= Better Steel + More Steel + Lower Costs

These are some of the Steel Manufacturers who use the Morgan-Isley Furnace Control System:

Bethlehem Steel Co., Various Plants	116
Armco Steel Corp., Various Plants	14
Sheffield Steel Co., Kansas City, Mo.	1
Sheffield Steel Co., Houston, Texas	8
Stanley Works, Bridgeport, Conn.	2
Dominion Foundries & Steel, Hamilton, Ontario, Canada	4
Heppenstall Co., Pittsburgh, Pa.	4
Empire Sheet & Tin Plate, Massillon, Ohio	1
Penn Steel Castings Co., Chester, Pa.	1
Altos Hornos de Mexico, Monclova, Mexico	3
Follansbee Steel Corp., Follansbee, W. Va.	1
Weirton Steel Corp., Weirton, W. Va.	

Write for complete information about the Morgan-Isley Furnace Control System

Morgan Construction Company
Worcester, Massachusetts

PITTSBURGH, P.A., 2815 Koppers Building
English Rep., International Construction Co., 56 Kingsway, London W.C. 2, Eng.

CCC-52

Who said it costs more

Here's where Stainless actually cuts cost 42.7%

COMPARISON AND

Cold Rolled Steel as against Stainless Steel

COLD ROLLED STEEL

Basic Cost of Tools. Experienced life of tools plus average cost of maintaining dies, pro-rated basis.	$.0152
Punch Press Operations	$.0180
Preparation for Chrome. Polish cold rolled before plating to remove "peel" and prepare finish to take clear color.	$.0224
Cost of Chrome Plating. including depreciation of equipment.	$.1331
Plating Scrap. Material scrapped because of prohibitive cost of stripping and re-chroming.	$.0140
Inspection	$.0060
Assembly line hold-ups. Costs over Time Study standard hour due to "built-up" pieces and chrome flashes interfering with assembly.	$.0230
Finished units rejected or returned. due to chipped or peeled plating resulting in "seconds" on line.	$.0480
Cost of material .025" Stock. less scrap value.	$.0121
Total	$.2918

to use Stainless Steel?

ANALYSIS OF COSTS

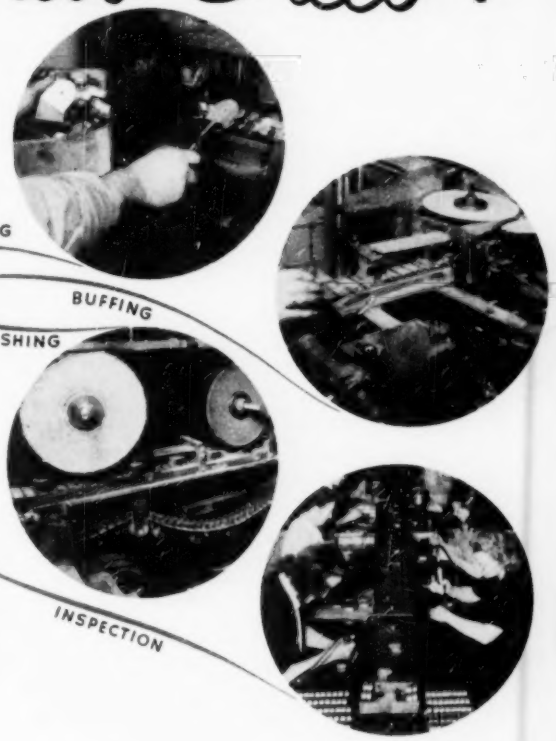
as experienced by Dart Mfg. Co., Mason, Mich.

STAINLESS STEEL

Basic Cost of Tools — Experienced life of tools plus average cost of maintaining dies, pro-rated basis.	**$.0334**
Punch Press Operations	**$.0180**
Polish and Buff — including depreciation of equipment and re-running of faulty buffing.	**$.0512**
Inspection	**$.0060**
Cost of Material .020" Stock — less scrap value.	**$.0584**
Total	**$.1670**

Cold Rolled Total Costs . . . . . $.2918
Stainless Total Costs 1670
Actual Experienced Savings . . . . . $.1248 per case

Percentage Reduction in Cost 42.7%



THE "Metro-Meter" is a little device you'll soon be seeing a lot more of. It's a dial-reading steel rule with built-in scribe that men just naturally "take to" because it's practical, obviously useful, and, encased in Stainless Steel, is permanently good looking.

It's the case we want to talk to you about. Originally it was made of cold rolled carbon steel, chrome plated. The steel, itself, was inexpensive. But by the time it was prepared for plating, and plated—after the costs for rejections and returns due to over-plating, under-plating, chipping and peeling were added —each case cost a little over 29 cents.

By using lustrous satin-finish Stainless Steel in place of cold rolled carbon steel, however, all plating costs as well as rejections and returns were eliminated. Even though the initial cost of the Stainless Steel used was about five times that of the carbon steel formerly used, the final *finished* cost in Stainless was exactly 16.7 cents per case—a saving of almost 12½ cents each, which means a 42.7% reduction in cost.

What is more, with the magic word "Stainless" now stamped on the case, an almost irresistible plus has been added. Sales for the "Metro-Meter" which formerly had been good, immediately became considerably better. Thus, Stainless has done here what it has done for many other products — improved appearance, increased durability and reduced sales resistance. And in this instance, it has actually reduced costs as well. In short, a product made of Stainless Steel does *not* necessarily cost more—it only looks as though it did.

We would be glad to have the opportunity to show you where U·S·S Stainless Steel can be applied to improve YOUR product — to increase its desirability — and, more often than you may expect, to reduce its cost as well.

AMERICAN STEEL & WIRE COMPANY, GENERAL OFFICES: CLEVELAND, OHIO
CARNEGIE-ILLINOIS STEEL CORPORATION, PITTSBURGH & CHICAGO
COLUMBIA STEEL COMPANY, SAN FRANCISCO · NATIONAL TUBE COMPANY, PITTSBURGH
TENNESSEE COAL, IRON & RAILROAD COMPANY, BIRMINGHAM
UNITED STATES STEEL SUPPLY COMPANY, WAREHOUSE DISTRIBUTORS, COAST-TO-COAST
UNITED STATES STEEL EXPORT COMPANY, NEW YORK

U·S·S STAINLESS STEEL

SHEETS · STRIP · PLATES · BARS · BILLETS
PIPE · TUBES · WIRE · SPECIAL SECTIONS

United States Steel Corporation Subsidiaries
Room 2061 Carnegie Building, Pittsburgh 30, Pa.

☐ Please send me the new book, "An Introduction to U·S·S Stainless Steel."
☐ Please have a Stainless representative call on me.

Name.....
Position.....
Company.....
Address.....
City..... Zone.... State.....

UNITED STATES STEEL

● Type HD-1430 Hevi Duty Vertical Retort Furnace at The Triplex Screw Company, Cleveland, Ohio.

it
COSTS LESS
to Case Harden
with
HEVI DUTY
Carburizing
FURNACES

Accuracy of Control and Uniformity of Electric Heat Enables this Company to *Produce Consistently Precise Results!*

● Here is a typical example of how a nationally known company has had excellent results with their new carburizing installation. Triplex says they have reduced case hardening costs . . . gained better control of heat treating quality. As a result, they have visible savings in heat treating expense, and the more intangible benefits of convenience, speed of production, and higher quality of products. You, too, can produce consistently precise results if you specify Hevi Duty Furnaces. Tell us about your problem . . . we can help you.

SEND FOR BULLETIN HD-646 TODAY!

HEVI DUTY ELECTRIC COMPANY

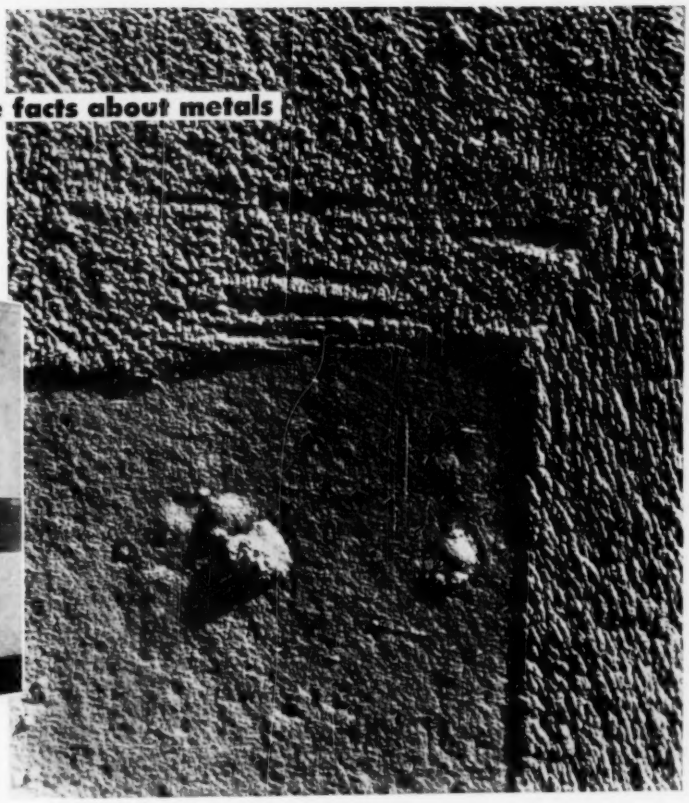
HEAT TREATING FURNACES **HEVI-DUTY** ELECTRIC EXCLUSIVELY

DRY TYPE TRANSFORMERS — CONSTANT CURRENT REGULATORS

MILWAUKEE 1, WISCONSIN

—for more facts about metals

The RCA type EMU Universal Electron Microscope. With it you can study particle details down to 1/2,500,000th of an inch.



Higher Magnification

—with the RCA Electron Microscope

This is an electron micrograph of stainless steel—electrolytically etched. It is magnified 22,000 times by the RCA Electron Microscope.

See how readily you can examine the grain structure in the large-grain areas. See how clearly the grain boundaries are revealed by the action of the etching on the surface of the steel.

Electron micrographs like this help the metallurgist uncover fundamental facts about crystal structure, corrosion, etching, abrasion, phase distribution of alloys, surface characteristics, and the like—because the high magnification of the Electron Microscope makes surface details visible to a degree not possible before. The RCA Electron Microscope produces sharp images at direct magnifications from 70 to 25,000 diameters. Photographic enlargements are possible up to 100,000 diameters—without exceeding the limit of the resolving power of the instrument.

The Electron Microscope enables the metallurgist to overcome the greatest single limitation of the light microscope—limited magnification. For in practically all cases, the electron microscope permits magnification of the specimen to the useful limit of the actual detail in the specimen itself. This greater magnification is the key to better understanding in metallurgy.

For helpful information about the RCA Electron Microscope, drop a card or letter to Dept. 130F, RCA Engineering Products, Camden, N. J.

SCIENTIFIC INSTRUMENTS

RADIO CORPORATION of AMERICA

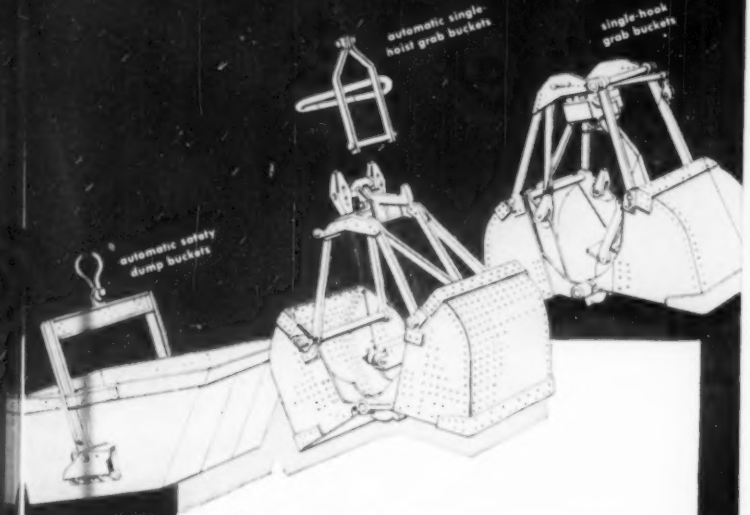
ENGINEERING PRODUCTS DEPARTMENT, CAMDEN, N.J.

In Canada: RCA VICTOR Company Limited, Montreal

dependable*

is a word to describe

BROSIUS BUCKETS



*Dependable means trustworthy or reliable, all of which aptly describes Brosius Buckets, and here's why: Brosius Buckets are sturdily built of heavy steel forgings, castings and plates and all parts are oversized for extra wear. Design has been purified to the point of maximum practicality and simplicity of operation. Many Brosius Buckets have been in operation for more than thirty years and they aren't ready for replacement yet. If you want a bucket you can depend on to suit your purpose and last your lifetime, write to Brosius for details and prices.

Edgar E. BROSIUS Company Inc.

Pittsburgh 30, Pa.

European representative: Brosius & Co., Geneva, Switzerland

CARBON and SULFUR ANALYSIS with Speed! Ease! Accuracy!

TWO MINUTE CARBON DETERMINATOR

High degree of accuracy assured by: fan cooling gases, leveling to meniscus line, automatically maintaining atmospheric pressure, wide selection of carbon percentage scales. Accurate within 0.002% on low range Burette.

THREE MINUTE SULFUR DETERMINATOR

Determination in accordance with A.S.T.M. specification accuracy within 0.002% inorganic or organic materials. Simple procedure.

HIGH-TEMPERATURE FURNACE

Sturdy construction with HIGHEST QUALITY insulation.

Use **3031D Boats and Zircotubes to reduce operating cost of analysis.**

Use Dietert-Detroit Carbon and Sulfur Determinators for Economical Analyses

Write to Dept. C-1 for descriptive literature

CONTROL EQUIPMENT

HARRY W. DIETERT COMPANY

9330 ROSELAWN • DETROIT 4, MICH.

Engineering Societies Service

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service, Inc., operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18;** 100 Farnsworth Ave., **Detroit;** 57 Post St., **San Francisco;** 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York Office and include 6¢ in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter or $12 a year.

Allen Joins Personnel Service

Bonnell H. Allen has been appointed assistant to the manager of the Chicago office of the Engineering Societies Personnel Service., Inc. Mr. Allen, a licensed engineer, has had a background of a number of years in engineering personnel placement work.

Positions Open

Cost Reduction Engineer with five to ten years' experience in cost reduction, methods and sheet metal forming, paint process and welding. Knowledge of tooling and jig design helpful. Salary, $5500 a year. Prefer resident of New Jersey. Y3041(a).

Engineers. (a) *Tool Room Foreman*, 35 to 50, experienced with tools, dies, progressive dies and eyelet machine tools that are used in the manufacture of brass fabricated items of small and medium design. Must be able to produce the tools on schedule and at estimated costs. Salary, $4000 to $4300 a year. (b) *Production Supervisor*, 35 to 50, experienced in the production of metal fabricated parts of small and medium design. Experience must be broad with all types of brass, eyelet machines and other metal working machinery. Must be able to produce parts at the estimated cost. Salary, $3600 to $3900 a year. (c) *Mechanical Engineer*, 35 to 50, technically trained in metallurgy and engineering. Must be able to cover the estimating, the designing and processing of brass fabricated parts. Work will involve brass fabricating of small and medium parts and he must have an intimate knowledge of tools, dies, progressive dies and eyelet machine tools. Salary open. Location, Connecticut. Y3363.

Assistant Instructor, chemical and metallurgical graduate, preferably with M.S. degree, to supervise laboratory courses and teach some general engineering subjects. Salary, $3220 a year. Location, New York Metropolitan Area. Y3396.

Production Superintendent, 30 to 40, mechanical or industrial graduate, with two to three years' experience in supervision, scheduling of work, time and methods procedures, for metal working industry. Salary, $4500 to $5000 a year. Location, Midwest. Y3430-C.

Production Engineer with at least 3 to 5 years' experience in the routing and control of production for small metal parts. Salary, $6000 a year. Location, New York, N. Y. Y3449.

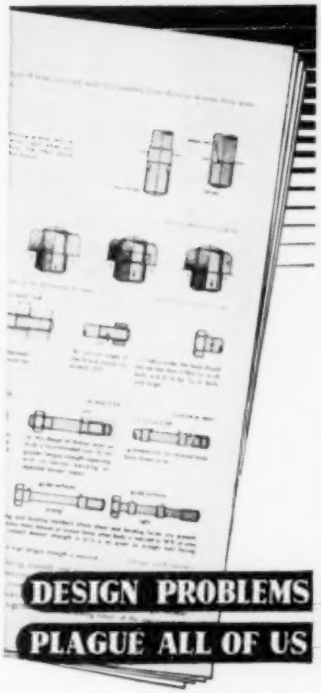
Engineers. (a) *Superintendent* with experience in erection shop for light structural and heavy sheet metal manufacturing. Shop employs 60 to 75 people. Salary, $5000 a year. (b) *Designer*, mechanical, to make own calculations, layout, calculations of cost and manufacturing on light structural and heavy sheet metal. Salary, $4200 a year. Location, Massachusetts. Y3470.

(Continued on next page)

TOOL AND DIE DESIGNER. For powder metallurgy parts. Experienced in commercial production and capable of directing tooling and press set-up. Location—Midwest. Salary—open. Give complete details of experience.

Box H-7—JOURNAL OF METALS

WANTED—Metallurgist for melting department specializing in the production of stainless, high-temperature alloys, and tool steels. Preference given to man having several years' experience with arc furnace production of above materials.

Box H-6—JOURNAL OF METALS

To serve well, a machine part—even when made of good steel, properly treated—must be properly designed.

A new 72 page booklet, sent free on request, discusses the vital relation between design, good steel and its satisfactory treatment.

Climax Molybdenum Company

500 Fifth Avenue
New York City

Please send your
FREE BOOKLET
3 KEYS TO SATISFACTION

Name.....
Position.....
Company.....
Address.....

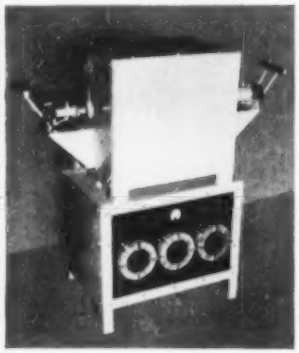
JM-5 F24

Harper

DESIGNERS AND BUILDERS
of
HIGH TEMPERATURE
Electric
FURNACES and KILNS

• *for:*

Assaying
Annealing
Batch Melting
Brazing
Ceramics
Heat Treating
Research
Sintering
Special Applications



Model GT-20-HM-30 Tube Furnace adaptable for rapid testing of small specimens. Maximum temperature 3000°F.

Write for data on standard models, and if your requirements are of a special nature, state the particulars and our engineers will be glad to design and quote on special equipment for you.

HARPER ELECTRIC
FURNACE CORPORATION
1456 Buffalo Avenue
NIAGARA FALLS, NEW YORK

PERSONNEL SERVICE

Sales Engineer, Metallurgical graduate, under 30, with good sales personality. Must have had some experience in the processes of steel melting in steel mills or foundries, particularly steel production or quality control. Will engage in sale and use of raw materials to steel industry for a manufacturer. Salary, approximately $350 per month plus expenses and benefits. Car furnished. Travel 50 pct. Location, Midwest. Headquarters, Chicago. R6245.

Men Available

Junior Metallurgist; available June, 1950, B.S. in Chem. Eng. Some knowledge of statistical quality control. Desires position in quality control or product development. Prefer East. M-525.

Materials-Metallurgical Engineer. Twelve years' diversified experience in the manufacture, fabrication and application of materials, embracing specifications, heat treatment, engineering properties, corrosion problems, investigationary work, coordination and liaison. Experienced literature research, abstracting and indexing technical literature, technical and report writing. Supervisory and consulting experience. Single, free to travel. Location immaterial. M-526.

Metallurgical Trainee, young, able, willing to learn. Will be available June, 1950, and will obtain degree of B.Ch.E. (Metallurgy option) from New York University. Knowledge of quality control. Desires position development, production, research or process control. Location immaterial. Single, will travel. M-529.

Metallurgical Engineer, M.S., 34, married. Seven years' varied experience all phases ferrous and non-ferrous metallurgical research and development scale-up in supervisory capacity. Qualified to act as technical assistant to busy executive. Presently employed; prefer U.S. M-519.

Physical Metallurgist, D. Eng., June, 1950. Thorough metallurgical education and research experience both in alloys and metal single crystals. Especially familiar with field of study on age-hardening, phase transformation, deformation, recrystallization and grain growth. Desires position research, development or teaching with opportunity to do research. Age 28, married. Location immaterial. M-520.

June Graduate, B.S. in Ch. E., Metallurgy major; desires position metallurgy development or foundry departments. Veteran, 25, single. Will work in U.S. M-532.

Metallurgical Engineer, B. Met. E. Several years' executive experience in production, research, development and application of both ferrous and non-ferrous metals, including fabrication of powder and precious metals. Desires position where knowledge and experience will lead to executive position in management. Presently employed, but desires change that will utilize full ability. M-533.

Metallurgical Engineer, Aviation Field. B.S. in Metallurgical Engineering. Have Commercial Pilot's License (single and multi-engine, land and/or sea); Aircraft & Aircraft Engine Mechanic's License; Aircraft & Aircraft Engine Instructor's License. Served as Naval and Coast Guard Aviator during war. Desire position utilizing training and experience in aviation and metallurgy. M-534.

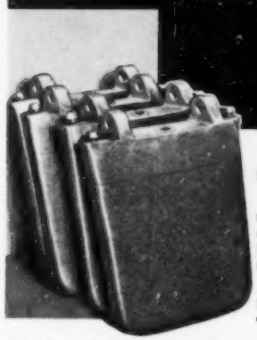
Metallurgical Engineer, 29, single. Graduate Ch. E., M.S. in Met. Eng., 1950. Practical experience and background in gray iron metallurgy. Desire position in research or development. Will consider instructorship, full or part time, with chance to work for doctorate. Prefer Midwest. M-535-293-Chicago.

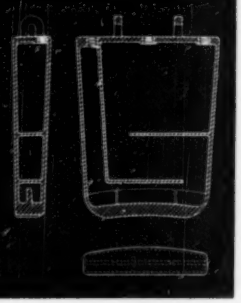
June Graduate, B.S. Chem. Eng., Metallurgy major, 22, single. Desires position metallurgy development or research or quality control, preferably in United States. M-536.

Metallurgist, 37, married. Broad research experience, both nonmetallic and sulfide flotation, including pilot plant. Operating experience large copper concentrator and leaching plant. Well qualified direct mineral dressing research laboratory, or assist in operating plant. Presently employed, available one month's notice. Any location considered. M-509-4910-E-7-San Francisco.

Metallurgist, 37, married, one child. Twelve years' experience cyanide leaching, and counter current decantation, three years' experience as refinery superintendent both silver and gold refineries using Miller system of parting. Three years' experience plant mechanic large crushing plant. Desires responsible position and opportunity. Speak Spanish. Prefer Latin America. M-515.

HEAVY NOSE SUPER-COOLED BOSH PLATES





Designed especially for the tuyere zone where plates are most apt to burn. The inch thick copper nose is Super-Cooled by fins extending into the water chamber. Increased resistance to heat effects important reduction in plate losses.

Three year test shows 20 to 1 greater tonnage over ordinary plates.

This substantial saving was demonstrated on two furnaces under operating conditions as nearly similar as possible. The comparison was made between ordinary plates and Heavy Nose Plates in the tuyere zones of the two furnaces.

In addition to the great reduction in Bosh Plates per ton of iron, fewer shutdowns to change plates effected important savings.

To keep away from trouble and avoid needless costs, use Heavy Nose Super-Cooled Bosh Plates in the tuyere zone rows.

Made by Smeeth-Harwood specialized melting and molding techniques, assuring dense metal of measured thickness.

SMEETH-HARWOOD COMPANY
2401-09 West Cermak Road, Chicago 8, Illinois

Superior* Blast Furnace Copper Castings *Exclusively.

POSITIONS AVAILABLE

Required for a new Electrolyte Copper refinery, men who have had previous copper refining experience. Positions available are: Assistant Superintendent, General Foreman and Assistant General Foreman in the Tank house and Electrolyte Purification section, Assistant General Foreman in the Precious Metal and By-Products department and various under foremen for not only these departments, but the Copper Casting section as well. In addition, employment is available for men experienced in sampling, inspection, and laboratory methods used in copper refineries. The plant is being erected in the Rocky Mountain area.

Complete details regarding past employment record, together with references, personal data, etc., to be furnished with applicant's first reply. Salary will be determined by experience and record of the individual. All correspondence will be considered absolutely confidential.

Box G-5—JOURNAL OF METALS

It's not so late as you think!
If you . . .
THINK INSTRUMENTAL-WISE with
BETTER ANALYSIS

BT

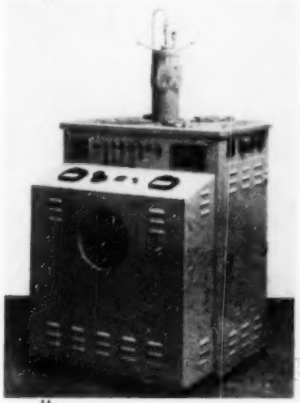
Baird Associates

- **3-METER GRATING SPECTROGRAPH**
  Descriptive Bulletin XXXII
- **DIRECT READING SPECTROMETER**
  Descriptive Bulletin XXVI
- **INFRARED SPECTROPHOTOMETER**
  New Descriptive Bulletin XXXIII

Precise instrumentation for the organic and inorganic chemist, for the metallurgical engineer, who know how to utilize savings. Technical bulletins for each of the above instruments are yours for your letterhead request.

Baird Associates Inc.
33 University Road Cambridge 38, Mass.

If you want a MODERN Diffraction Unit



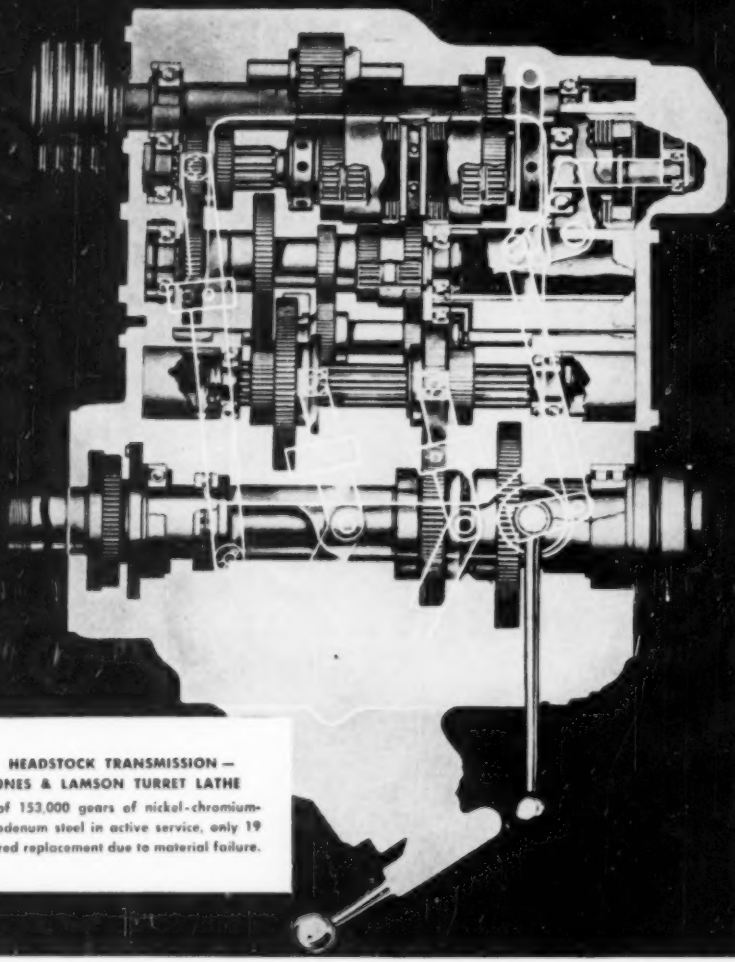
- A unit of tremendous x-ray intensity—exposure times reduced to ¼ that required by units of conventional design.
- A unit of unusual flexibility—4 equal intensity line focus radiations from as many as 4 different targets available simultaneously. Targets may be changed while the unit is operating.
- A unit of amazing operating economy—expensive replacement tubes are never required.

then specify **THE HILGER X-RAY DIFFRACTION UNIT**

For a complete description of this very flexible instrument, write for the FREE bulletin 304

JARRELL-ASH COMPANY
165 NEWBURG STREET **BOSTON 16, MASS.**

CAN YOUR EQUIPMENT MATCH THIS AMAZING RECORD?



HEADSTOCK TRANSMISSION — JONES & LAMSON TURRET LATHE

Out of 153,000 gears of nickel-chromium-molybdenum steel in active service, only 19 required replacement due to material failure.

A study of replacements during a 12-month period of World War II revealed an astonishing performance record for nickel-chromium-molybdenum steel gears used in the headstocks of Jones & Lamson Ram Type Turret Lathes. In spite of frequent overloads and punishment due to inexperienced operators, failures were cut to less than 00.02%. Because of this remarkable record, Jones & Lamson Machine Co., Springfield, Vt., continues to specify these shafts and gears in "872" and "874" nickel-chromium-molybdenum steels produced by Carpenter Steel Company, Reading, Pa.

THE INTERNATIONAL NICKEL COMPANY, INC. 67 WALL STREET NEW YORK 5, N. Y.

* An electrolytic method of making wire as fine as 0.00015 in. diam has been developed by Armour Research Foundation. The wire is made an electrode in an electropolishing bath and is dissolved away to the desired size. The wire is designed for smaller, more compact electronic equipment, and scientific equipment may be made more sensitive with such extremely thin wire. Further work will include investigations of the adaptability of various metals to the process and the ultimate practical reduction of wire diameter.

* Salt bath furnaces are being used for removal of burnt-in sand from castings. Cored holes, undercuts and deep recesses in castings are completely desanded when immersed in special salt baths operating at 700°F. The salt bath does not attack the metal no matter how long it remains in the bath. Chevrolet-Cleveland is using the technique on automatic transmission castings.

* In British bessemer practice, nitrogen and phosphorus in steel are reduced to about half normal values by a combination of thin brick linings that permits a shallower bath of iron, shortening the path of air blown through the metal; and the addition of mill scale in place of steel scrap to cause direct oxidation of the charge and prevent oxygen in the air blast from being used up.

* 460,000 metric tons of iron ore from the Philippines will have been shipped to Japan by the end of June. Used in the ore transport are 36 Japanese ships operating under American supervision.

* Unprecedented income and profits taxes plus a sharp increase in building costs have worked havoc on new ship construction in Great Britain. Shipping lost in the war were replaced ton for ton since the war, helped by war risks insurance. It is estimated that for a deep sea tramp ship, the cost in 1945 was 169 pct and in 1949 was 254 pct of the 1939 figure.

* Radioactive tracing techniques have shown that when two pieces of metal are rubbed together, atoms of one transfer and become part of the other. Traces of metal from piston rings in a gasoline engine were detected forming part of the cylinder wall, even with excellent lubricants. The only way to remove the radioactivity of the rubbed metal is by honing, which gives an idea of the bond. The technique might be quite useful in evaluating lubricant effectiveness.

* French steelmakers are taking the leadership in international competition, offering steel products at reduced prices. Merchant bars are reported selling for $48 per ton f.o.b. in quantities of 500 tons or more. Belgian and German prices are about 8 pct and 12 pct higher, respectively with the Germans offering to refund the difference on large orders. German deliveries are prompt. Some hesitation has been found in placing orders with French producers because of irregularity of dates of shipment and the fear of the political situation and strikes.

* A new open hearth scrap handling and charging technique, which permits charging one buggy of scrap every 2 min., is being built into the No. 2 open hearth shop at the Armco Steel Corp. at Middletown, Ohio. A novel method of delivering loaded buggies, and removing empty buggies from the charging floor, is the key to the Armco scheme. (p. 744)

* A modernization program in the field of blast furnace gas cleaning is now under way at the Gary Works of the Carnegie-Illinois Steel Corp. Two plate-type electrical precipitator units and three high-efficiency boilers to fine-clean and burn the gas have been installed. Tube-type electrical precipitators are also being added. Ultimate replacement of all Theissen washer houses is expected. (p. 746)

It's Everyone's Business

APRIL 10—Officially, spring comes to the Great Lakes on March 21 as it does elsewhere in the country but in the Superior district continued snow and freezing until late in March have caused citizens in Duluth to place their bets on April 15 for the first ore vessels if the ice breakup is left to Mother Nature.

North and west of Duluth and near the eastern end of the Mesabi lies the little town of Aurora, population 1100. As the little v-nosed snowplow, pride of the community, shuttled back and forth clearing Aurora's few sidewalks, the crowd in Bradich's Cafe were more interested in the impending high school state championship basketball tournament in Minneapolis than in the outcome of Erie Mining Company's petition for water rights for their proposed 10 million-ton taconite concentrating plant. Such a plant would employ 4500 workmen, and at the prevailing ratio of 10 people for every mine worker in Minnesota, this would mean a 45,000 increase in population for Aurora and the vicinity in the next few years.

Downstate fishermen are alarmed at the flurry of activity by the mining companies in pushing plans for large scale recovery of iron from taconite. Inconvenience to these gentlemen will be inconsequential when compared with the vast increase in payrolls for range communities. Oliver is pushing the construction of a fine ore agglomeration plant at Virginia which will give them valuable experience for projected taconite processing. Oglebay, Norton & Co. is ready to go at Beaver Bay, north of Duluth, as the right of way for a railroad from Babitt to Beaver Bay is secured as well as permission to dispose of tailing into Lake Superior at the site of the proposed concentrating plant. This company is also doing research in agglomeration at Ashland, Ky., in cooperation with other interests. The financing is yet to be worked out.

Of the four possibilities for agglomerating taconite concentrate—sintering, pellitizing, nodulizing and briquetting—the prospects for pelletizing appear the most favorable. It seems likely that 60 cents per ton can be saved by this process over sintering because only 1 to 2 pct of coal is required as compared with 7 pct for sintering. One reason for the lower coal consumption is that the chemical transition of magnetite to hematite which transpires in the pelletizing process is exothermic.

Foreign policy has become a matter of partisan politics in recent months partly because of Senator McCarthy's attack on the State Department, the loss of China to the Communists, and the illness of Senator Vandenberg, chief architect of bipartisan foreign policy. Steps have been taken for a reaffirmation of bipartisanship by the appointment of two Republicans, John Foster Dulles, a former Senator, and John Sherman Cooper, former Senator from Kentucky, as "consultants" to Secretary of State Dean Acheson. Meanwhile, Senator Vandenberg saved his party's blushes by neither condemning it for obstructing the European recovery program nor rolling the State Department in the mud. He simply wrote a letter to another Republican, Paul G. Hoffman, ECA head, in which he reminds his colleagues that they are engaged in a struggle that is epochal in nature and of global design. He pleaded for a return to unity and suggested that since American commitments extend beyond the election year 1952, a nonpartisan commission composed of the country's ablest men should consider the best way of meeting them.

There are strong political pressures that could conceivably block this effort. For instance, Senator Robert A. Taft has a different definition of bipartisanship than that of the Administration. He thinks that Republicans in Congress should be consulted rather than the Administration's hand-picked "consultants". It is true, however, that Vandenberg recommended the selection of Dulles as a "consultant" to Secretary Acheson.

It is to be hoped that unity will prevail as the problems to be faced are sufficiently knotty to require the best heads in our government, be they Democrat or Republican. In the Far East the further spread of Communism must be stopped, and in Western Europe the dollar gap must be closed. These will be difficult feats after the Marshall Plan ends in 1952.

Senator Vandenberg's letter to Mr. Hoffman was taken in certain foreign quarters to be an admission that American economic aid would continue beyond 1952. Considering that the best estimates made by OEEC indicate a deficit of $1 billion for Western Europe and its dependent territories in the first year beyond 1952, this is

not surprising. Besides offsetting this dollar gap by increased importation of foreign commodities, it will be necessary to encourage the investment of American private capital abroad. Exploitation of foreign sources by American oil companies are a classic example of this type of investment. The London Economist suggests that if President Truman's Point IV program is to overcome the dollar gap by mobilization of American capital for this type of investment, private savings and subscriptions to public issues by the private investor through an investment corporation in which the transfer risks are guaranteed by the U. S. Government will have to be made.

ECA negotiated $54,745,580 in purchases for the U. S. stockpile in the first two years of its operation. The purchases were made by utilizing the U. S. portion of counterpart funds deposited by European countries to match aid furnished under the Marshall Plan. ECA contracts for purchasing for the stockpile have enabled certain foreign mining ventures to come into or expand production. Besides supplying our stockpile, they have made additional sources of supply for metal consuming industries here.

Although the stockpile represents military security to a degree, foreign sources of supply would be tenuous in the event of war. American strategic mineral development, excepting uranium, has been allowed to lag for the lack of government support. J. Carson Adkerson, of the American Manganese Producers Association, urged approval of a bill authorizing the Government to underwrite domestic production of low-grade manganese ores. Such a program would require government price support and long term contracts. Although the Bureau of Mines and the USGS have done considerable work in finding and estimating the grade of many strategic mineral deposits, they cannot be brought into production rapidly enough to meet wartime contingencies.

Rising living costs have taken away the share of goods to which old-age pensioners and other fixed-income persons are entitled. For many in this group the decline in interest rates on bonds and mortgages, together with higher prices, have created a double squeeze on real purchasing power. The accompanying chart shows the relationship between average hourly earnings, productivity in terms of output per manhour, and prices of manufactured goods. A striking example of the reason for the rising cost of living is shown by the increase of average hourly earnings in recent years without a corresponding increase in output per manhour. Although these figures are approximate, the decade between 1939 and 1949 saw a 120 pct increase in average hourly earnings while manhour output has probably gained but 15 pct. Prices have therefore been forced up almost 90 pct. The average hourly earning figure does not include labor costs for vacations, sickness, pensions, etc. The productivity figures are rough because of changing composition and quality of products made, and for this reason no figures are given for the war period.

The other major influence in the declining purchasing power of the dollar is the great increase in government expenditures and the method of financing them. In the 1920's all levels of government absorbed 8 pct of the available resources of men and materials but this figure has risen to over 18 pct today. Thus the government, by preempting its share of the resources in a period of already heavy demand, drives prices higher. The way to halt the spiral of increasing prices is to hold wage increases to a more reasonable relationship with productivity and to reduce the draft on resources absorbed by big government.

Discussion of unemployment in Congressional committees has been largely theoretical and statistical; but with one out of every thirteen workers looking for a job, unemployment figures are beginning to make news. Although 58.5 million people are now drawing wages or salaries, nearly 4.7 million people were unemployed in the week ending February 11, and the number who had

been without work for 15 weeks or more rose by 160,000 between January and February, to a total of 1.1 million. The average length of time an unemployed person had been seeking work increased from eleven to twelve weeks. President Truman called for 61 million civilian jobs in 1950 and national income of $300 billion a year by 1955, all of which may make it rather difficult for the country to act like a creditor nation as efforts are intensified in this direction toward the latter part of the year.

The position in which British Overseas Airways Corp. has been placed by the late delivery of 14 Boeing stratocruisers points out the proximity of commercial gas turbine aircraft. BOAC ordered the aircraft for delivery in December, 1947 but the first did not arrive until October 15, 1949, and the last one was delivered the first week of April. Since the operating life of an aircraft is eight years, these planes should be run until 1958, but the turbo-prop Bristol 175 being developed by the Bristol Aeroplane Co. may possibly be ready in four years. Since gas turbine aircraft have lower running costs and added passenger comfort because of silent, vibrationless operation, it is questionable if the piston engine stratocruiser could compete on the same routes. BOAC is not the only airline faced with this problem of obsolescence of equipment before the expiration of its working life.

aime NEWS

1951 Officers Announced, Also Committeemen, Bylaw Change, Publications, at Mar. 15 Meeting

Announcement was made at the meeting of the Executive and Finance Committees of the AIME, at their meeting at the Engineers' Club, New York, on March 15, of the official ticket for Institute officers for 1951. W. M. Peirce heads the ticket, as the nominee for President. Mr. Peirce is chief of the research division of The New Jersey Zinc Co., at Palmerton, Pa. He is a member of the Institute of Metals Division of the Institute and has long been active in its executive councils. He served as a Director from 1937 to 1940, and thereafter was Vice-President for three years. Last year he again began serving a three-year term as Director, and is currently a member of the Executive Committee.

For Vice-Presidents, Joseph L. Gillson and Michael L. Haider were nominated, representing the Mining and Petroleum Branches respectively. Directors named were: Thomas G. Moore, Charles E. Lawall, John F. Myers, Fay W. Libbey, and A. C. Rubel. The Nominating Committee, O. B. J. Fraser chairman, met during the Annual Meeting. No letter ballot will be necessary unless 25 or more members offer a different complete or partial ticket before Sept. 1. Brief biographical sketches of each official nominee will be published in the July issues of the journals.

The Committees considered the request of the Council of the new Mining Branch of the Institute that part-time Eastern and Western paid secretaries be named to handle its affairs. No action was taken on the Western secretary, but approval was given of the diversion of half of John V. Beall's time, as editor of Mining Engineering, to secretarial duties for the Mining Branch. To assist him on the monthly journal a technically trained assistant editor was authorized. This man, when secured, will also assist in the editing of the Journal of Metals, of which Thomas E. Lloyd has succeeded Thomas W. Lippert as editor.

Provided the Admissions Committee approves, the Executive and Finance Committees looked with favor on a recommendation from the Council of Section Delegates that special consideration should be given to Junior Members transferring to a higher grade of membership. Upon reaching the age of 33 they are, according to the present bylaws, obliged to pay a $20 initiation fee and $20 annual dues as Associate Member or Member. The Section Delegates recommended that $1 per year of continuous dues-paying membership as a Student Associate or Junior Member be credited on the initiation fee, up to a maximum of $10. A change in the bylaws to effect this will be voted upon by the Board at its meeting on June 22.

In addition to the awards already authorized for presentation at the Annual Meeting in 1951, the Committees authorized the third award of the Erskine Ramsey Medal. This was made possible by a gift of $500 from Paul Weir, 1950 medalist, as an addition to the income of the Medal Fund.

Ross J. Leisk was named as chairman of the auxiliary publications committee of the Mining Subdivision; Richard H. Hunt, of the Geology Subdivision; and H. LeRoy Scharon, of the Geophysics Subdivision. The duties of these committees will be to review technical papers offered for Institute publication in their respective fields and make recommendations to the AIME Technical Publications Committee as to acceptance. As to the last-named committee, Francis B. Foley was appointed chairman to succeed E. C. Meagher, resigned. Lloyd C. Gibson was appointed to the Admissions Committee, succeeding T. B. Counselman, resigned.

Microfilming of current volumes of the three AIME journals by University Microfilms, Ann Arbor, Mich., was approved. This concern will offer these microfilms for sale to those who wish to preserve Institute publications in this way instead of the more bulky volumes as printed. C. H. Mathewson was authorized to begin preparation of a revised edition of "Modern Uses of Nonferrous Metals", a Seeley W. Mudd Memorial Fund volume. E. R. Kaiser was reappointed to serve as AIME representative on the Engineering Foundation board for a term of four years. The scheduled Board meeting for Wednesday, June 21, was postponed until the afternoon of Thursday, June 22, to be followed by a Directors' dinner at the Engineers' Club at which the Rand Medal for 1950 will be presented to Francis H. Brownell.

A memorandum from the Institute staff suggesting a more economical method of handling AIME publications was approved. Heretofore members have been allowed a special price on most Institute publications provided they purchase only one copy; if more than one they must pay the nonmember price. This required detailed records of members' purchases, checking of all members' orders, and often considerable correspondence. The suggested plan is to rescale the nonmember prices of all Institute publications except the journals, and to allow members and libraries a 30% discount from these prices, and dealers a 20% discount, with no limit as to the number purchased. The new price list will be offered for

Board approval at the April 21 meeting. On Transactions volumes a 50% discount will be allowed if members order the volume when paying their dues.

Earle E. Schumacher, vice-chairman, presided at the meeting, with E. C. Meagher and W. M. Peirce, of the Executive Committee, present, and Erle V. Daveler, A. B. Kinzel, and Gail F. Moulton, of the Finance Committee, and Andrew Fletcher, Treasurer, and Edward H. Robie, Secretary.

Proposed Amendment to Bylaws

The Board of Directors, at its meeting on February 13, 1950, gave notice of a further amendment to Article I, Section 5, of the Bylaws to be voted on at the April meeting of the Board, as follows:

Substitute for the second and third sentences the following: He shall be not more than thirty years of age at the time of his application, and shall not remain a Junior Member beyond his thirty-third birthday, except that the Board of Directors shall have the power to waive the age limitation in the case of veterans of World War II.

Proposed Amendment to Bylaws

Notice is hereby given that at the meeting of the Board of Directors on June 22, 1950, the following addition to Article II, Section 1, of the Bylaws will be voted upon (the new material is in italics):

"Each newly elected Member or Associate Member, and each Junior Member at the time of his transfer into the class of Member or Associate Member, shall pay, immediately upon notification of such election or such transfer, an initiation fee of $20; provided, however, that said initiation fee may be payable, at the option of the applicant, in four equal annual installments, the first of which shall be due upon notification of election or transfer; *and provided further that $1 per year of continuous dues-paying membership either as Student Associate or Junior Member or both, with a maximum of $10, be credited towards the initiation fee of Junior Members transferring to Associate Member or Member.* There shall be no initiation fee for Honorary Members or Junior Members."

AIME Transactions Volumes

All Transactions Volumes authorized for 1948 and 1949 publication are now available except Vol. 182, on Nonferrous Smelting and Refining, which should be published shortly. Copies have been sent to all those on our records as having ordered these volumes; others may place orders at $3 for paid-up members, $6 for nonmembers (domestic) and $6.50 (foreign).

The recent volumes include:

175. Institute of Metals (924 p.); 176. Iron and Steel (552 p.); 177. Coal (594 p.); 178. Mining Geology (556 p.); 179. Petroleum Development and Technology (395 p.); 180. Institute of Metals (713 p.); 181. Metal Mining, Industrial Minerals, Mineral Industry Education, and Geophysics. (Rocky Mountain Fund volume). (472 p.); 182. Nonferrous Smelting and Refining (330 p.) (available shortly); 183. Minerals Beneficiation (Gemmell Memorial Fund volume). (372 p.).

Transactions volumes in the new size, 8½ by 11½ in., covering all Transactions material published in the three journals in 1949, are now being issued. Vol. 186, Petroleum Development and Technology (332 p.) is already available. Soon to be issued are: Vol. 184, Mining Branch, and Vol. 185, Metals Branch. The price is $3.50 for a first copy to members, $7 to nonmembers in the United States, and $7.50 to nonmembers in foreign countries.

"Statistics of Oil and Gas Production", with data for 1948 (524 p.) was published last fall. It was supplied free to members who received the Journal of Petroleum Technology. To other members the price is $3, and to nonmembers $6 domestic and $6.50 foreign. Similar data for 1949 will be included in the Journal of Petroleum Technology during 1950, but a bound volume of this same material will be available at the end of 1950 or early in 1951, consisting of reprints of the statistical material in various issues of the Journal of Petroleum Technology.

E. E. Schumacher to Deliver Lecture

Earle E. Schumacher, chief metallurgist of Bell Telephone Laboratories, Inc., Murray Hill, N. J., has been invited to present the Fall Lecture of the Institute of Metals (London). The lecture will be presented at the Institute's meeting at Bournemouth, England, during the week of Sept. 15.

Earle E. Schumacher

The Institute holds three main general meetings a year, at which papers on scientific and technical subjects are read and discussed. At the fall meeting, a special lecture is given on a subject of particular interest to the area in which the meeting is held.

Following the meeting Mr. Schumacher plans to visit college, institute, and industrial laboratories in England, Holland, France, and Switzerland.

The *Drift of Things* . . . . . as followed by *Edward H. Robie*

Applicants for Membership

To what extent, if any, the names, addresses, biographical records, and endorsers of applicants for membership in the AIME should be published in the monthly journals has long been a matter of controversy. When we joined the Institute back in 1919, each applicant got an inch or two of space in the journal and all of the above information was included. As the years have gone by, less and less space has been given to this material, so that published data has recently been restricted to the name of the applicant, the town and state where he lives, and a code letter to indicate the grade of membership for which he has applied. Some feel that inasmuch as few read this matter it should be omitted entirely, thus saving the money required to publish it, or enabling the space to be utilized for more acceptable copy. The principal arguments for continuing to publish the names of applicants, in some form, are: (1) occasionally members notice the name of an undesirable candidate and are thus enabled to make a protest before the Board would otherwise accept the applicant; (2) members are thus enabled to see what new members have applied from their local territory; and (3) the applicant himself likes to see his name in print as having applied for membership in his professional society. For these reasons the Council of Section Delegates in February opposed complete omission of the names.

At its meeting on April 21 it seemed likely that the Board would accept a recommendation of the Admissions Committee that hereafter the following procedure be followed: The Branch affiliation of each applicant will be supplied to the respective Editors, who presumably will publish only the names of applicants in the Branch served by their respective journals.

This should make the list of more interest. Members are urged to peruse the names and to report to the Secretary's office if they note any who, they believe, for any reason should not become members, or who are applying for admission as Member when they should properly be Junior Members or Associate Members.

Handling Grievances

Complaints about poor service given members by Institute headquarters have dwindled to a small number but occasionally are still received. Doubtless others are annoyed by something headquarters has done or has not done but do not take the trouble to write in about it. Some members feel that even after they have registered a complaint matters are not adjusted to their satisfaction, and they thus have a just grievance.

To handle such cases, the Gillson Committee on Democratization suggested that an individual or a committee be appointed to investigate grievances and see that a satisfactory adjustment is made. The Board referred the suggestion to the Council of Section Delegates, which decided that its own membership should accept this responsibility. Therefore, a member who has a grievance should notify the representative of his own Local Section on the Council, or the Secretary of the Local Section if the Council member's name is unknown to him, and he will see that so far as possible the disgruntled member is satisfied.

The Numbers Racket

In 1949, ME and JM were each marked Vol. 1 because they were the first issues of two new magazines. However, as the Transactions papers therein carried different volume designations, it was decided in 1950 to give the magazine the same volume number as the Transactions volume would have. This will avoid the necessity for dual reference designation—that is, one reference for the magazine and one for the Transactions volume. JPT is, however, continuing with the same volume numbering as used in 1949.

Subsidies and Price Supports

The extent to which the taxpayers of the country are being mulcted to assure profits to various groups of producers in the United States has almost assumed the proportions of a national scandal, and fortunately is receiving increasing attention by our legislators in Washington. A good part of the deficit in our national budget could probably be ended if the Government would stop paying subsidies or buying up products for which there is no market. Likewise the consumer—the public—would be able to purchase these goods for a much more reasonable price.

Most of this Government money goes to agricultural or similar products of the land. Such products are largely the type that, unlike the products of the mineral industry, will deteriorate in storage and thus must be disposed of, usually for little or nothing, or allowed to spoil. In general they must not be disposed of for what they will bring in the market for that would defeat the purpose of the legislation.

On March 1, 1950, the value of the farm crops that the Government had on hand was $4,036,175,453 and the Department of Agriculture has requested that two billion dollars more be appropriated to take care of commitments and guarantees the Government has already made. The Government holds more than a billion dollars' worth each of wheat and corn, for instance; almost a billion dollars' worth of cotton; $57,000,000 worth of butter.

Though the situation is anything but funny a rather amusing item appeared in the *New York Herald Tribune* on March 7, which said: "Officials said the department now has about 121,000,000 pounds of dried milk and 72,000,000 pounds of dried eggs. About 1,000,000 pounds of fried eggs have been given away to public welfare groups." They apparently not only give away the eggs but they actually fry them for the beneficiaries, some of whom would probably rather have them scrambled.

Though some in the mining industry have clamored in the past for more Government aid, it is a matter of considerable satisfaction to

many that the mining industry is not open to censure in the present sorry mess.

This challenge to the American free enterprise system, and interference with the old law of demand and supply, is also indulged in by many manufacturers who decry the Government's action as to farm commodities. Through the price-fixing laws—euphemistically called "fair-trade" laws—in 45 of our States (but not in Texas, God bless them) a retailer may not cut the price below that set by the manufacturer on a host of articles that he sells. The law is being violated by an increasing number of sellers, and Macy's department store in New York has the matter in the courts. This practice is different of course from Government subsidy and price support, but similar in that the consumer is often forced to pay more than he would in a free market.

Everybody is organized into pressure groups except the consumer.

Coal Still an Industry

Higher prices for coal resulting from the recent strike are said to have led to further inroads by the oil and gas producers on the market for the solid fuel. Some miners are not getting the increased wages they expected because their jobs have evaporated completely. But the coal industry is neither dying nor dead. Witness this extract from the annual report of the Pittsburgh Consolidation Coal Co.: "With all the chaos in the coal industry in 1949 the bituminous mines and the men employed in them produced some 430 million tons, or slightly less than three tons for every inhabitant in the country. This tonnage is about equal to 25% of the combined production of the world and is almost twice the amount of coal produced in any other single country. Oil, gas, and coal combined permit this country to use approximately seven times as much energy per person as is used in the world as a whole. However, price and dependability will determine to a large extent how much of the total energy will be supplied by each of these fuels. In spite of the fact that the productivity of our American miner is from three to six times greater than that of the miners in foreign countries, we must make even greater strides in this regard so that the cost will not bar coal from enjoying its proper share of our increasing national use of energy."

When we consider the enormous resources of the United States, and its immense advantage over all other countries in its use of energy, we are optimistic that Russia will never have the temerity to engage in a war with this country. The only reason why we get so worried sometimes is because we are not sure that our optimism is justified.

Man's Best Friend

A dog, it has been said, is man's best friend. Especially is a friend appreciated when he occasionally serves utilitarian purposes. Thus do the dogs of the North pull sleds; others are excellent guardians; some guide the blind. A few carry whiskey to mountaineers in the Alps, lost in the snow, if we are to believe the famous English animal painter, Sir Edwin Landseer. Many dogs are good hunters. Not all, though, for we once asked a friend how it happened that his dog was lame and his rather sneering reply was, "a rabbit must've bit him."

Getting back to the mineral industry, which we usually do sooner or later, some of our most famous mines have been discovered by dogs. The story usually goes that the old prospector is about ready to give up when he notices his faithful hound is scratching violently not far from camp. Investigating, he finds that he has uncovered gleaming specimens of high-grade ore, and thus is another grass-roots bonanza launched.

But not until we read a story in the *Knoxville Journal* of March 19 headed, "U-T Geology Student Has Wonderful Assistant In Gifted Mongrel Dog", did we realize what an aid a dog could be to a geologist. We do not wish to give the impression that the *Knoxville Journal* is our regular daily reading fare: we are referring to a clipping sent us by the Drift's scout in Oak Ridge, Ed Miller. It seems that one of the University of Tennessee geology students found a mongrel pup to which he took a liking, who accompanied him on his field trips. Until he was about a year old, the dog behaved like any other canine. But one day the student was collecting fragments of trilobites when he noticed that his companion was taking a keen interest in the fossils and was digging into the bank with his forepaws. He found several specimens in good shape and carried them in his mouth to the collector.

As he became more adept, a new name for the dog seemed in order, and we were particularly fascinated by the one chosen: "Coprolites". For the benefit of readers of the Drift whose education has been incomplete we refer to the dictionary: Coprolite—a roundish, stony mass consisting of petrified fecal matter of animals.

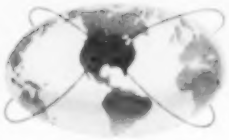
Coprolites has been with his master on many trips and has developed a distinctive bark for each of the various types of fossil remains. For a crinoid he gives a long, high yowl, but for an anemone, a series of short yelps. A pelecypod brings low growls; an ostracod discovery is marked by a baritone wail. Coprolites favors shale outcrops for they are easy to dig, but he occasionally can extract weathered remains from sandstone and limestone.

The student, who for some reason declined to have his name used, admits that he does not understand how the dog locates the fossils unless it is done purely by visual means. There is no longer any smell to go with the silicified remains. The student says there is nothing prettier in the world than to hear old Coprolites when he lets out a long wolf howl after finding a particularly rare fossil.

Only once has the faithful dog been fooled. He wandered into a cave and found a cow skeleton and was so excited he returned to his master, grabbed him by the sleeve, and almost dragged him into the cave. The student did not hold this one mistake against Coprolites as it was quite evident that the cow had been dead a long, long time.

Harvard and Yale are said to have been trying to purchase Coprolites for several months, as a field assistant in the geology departments.

So far, Coprolites has not applied for Student Associateship in the AIME. But as an associate of a student, and a most valuable one too, one might think he would logically qualify.



SECTION ACTIVITIES

Buffalo Section, OHC

The inaugural meeting of the Buffalo Section of the Open Hearth Steel Committee was held Thursday, Feb. 23. About 50 guests made the trip through the Bethlehem open hearth in the morning.

A total registration of 91 of the area's operators and technical men attended the afternoon operating session and stayed for dinner, with all of the activities at the Bethlehem Supervisors' Club.

H. L. Allen, Jr.

The chairman of the Buffalo Section presided at the technical session which was divided into the following subjects: Practical aspects of Bath Temperature Control—a discussion led by Jim Creighton, superintendent of steel production at Bethlehem; General Bottom Maintenance, led by Keith Moore, superintendent of open hearths, Steel Co. of Canada; and Pit Practice, led by Harry Northrup, superintendent of open hearths at Republic Steel Corp.

Registration represented nine companies in the area. The National Office was represented by Ernest Kirkendall.

Chicago Section

Prospective developments in steel making during the next decade was the subject discussed at the Feb. 1 meeting of the Chicago Section. R. A. Lindgren, technical chairman, introduced the two speakers, H. W. Johnson, of Inland Steel, and C. D. King, of U. S. Steel.

Mr. Johnson discussed the need for additional steelmaking capacity. Because of the high cost of construction, every effort is being made to improve present processes in order to meet increased demand. He spoke of means of improving existing capacities by better techniques, and new technological advances, together with the likelihood of present processes being replaced by entirely new developments.

Mr. King dealt with operating production problems, illustrating the present status of conventional open-hearth practice, and pointed out some of the deficiencies that will command intensive attention in the future, and the form that the open-hearth process of the future may take. He also touched on the less conventional duplex process as well as some expected trends in Bessemer and electric furnace production.

A record attendance was scored for a Chicago Section meeting with 256 present.

"Recent Developments in Powder Metallurgy" was the subject discussed at the March 1 meeting of the Chicago Section of the AIME. J. E. Drapeau, Jr., technical director of Metals Refining Co. Division of Glidden Co., was the technical chairman, and introduced the speaker, Dr. Frederick N. Rhines, professor, Metallurgical Engineering, Carnegie Institute of Technology.

Dr. Rhines' talk dealt with the substantial advances that have been made in recent years in the knowledge of the how and why of the sintering of metals. He pointed out that recent research in this field has contributed not only to the understanding of the sintering process, as such, but also, in a general way, to the understanding of the fundamental nature of metals. His talk was supplemented with slides. A lively discussion period followed.

Cleveland Section

W. T. Rogers, metallurgical statistician for the National Tube Co. plant at Lorain, Ohio, presented a talk on "Statistical Methods for Quality Control in Metallurgical Work" at the Feb. 23 meeting. The subject matter was timely and was most interesting to those listeners whose plants have already started on quality control programs.

Mr. Rogers described how in the late '20s German metallurgical technicians began to use these methods successfully. National Tube executives became interested and instituted a minor program in the early '30s. Their metallurgical department now has its own statistical division consisting of six people who daily correlate all technical information obtained by the chemical and the steel quality divisions in their observations and tests on mill products.

IBM system facilitates correlating the data. Correlated data is printed and distributed daily, weekly, and monthly. Mr. Rogers illustrated how these correlated data are used to solve plant metallurgical problems, citing several specific examples.

Southwestern Section, OHC

Substitutes for Pig Iron and Its Effect on Production was a timely paper to start off the Southwestern Section, NOHC, St. Louis meeting on Mar. 7, especially with a coal strike going on and the large differential between the costs of pig iron and No. 1 heavy melting steel scrap. Although quality is affected to some extent because of lower control of residual scrap alloys such as copper and nickel, the higher rate of production will not justify the use of a normal pig iron charge, even though the heat time would be considerably less than heats in which coke or graphite were used. Steel foundries and nonintegrated steel plants agree that fewer No. 2 bundles and No. 2 heavy melt can be used in heats using graphite and coke, if the same quality is to be maintained.

Sessions at the Lennox Hotel on that March day attracted 96 persons to morning and afternoon technical papers on six major topics of special interest to nonintegrated steel plants, and to the fellowship dinner. During the meeting, Clarence King, open hearth superintendent at Sheffield Steel, was elected chairman and Gordon McMillin, metallurgist for General Steel Castings, was elected vice-chairman of the section for the next two years. A. H. Sommer is the retiring chairman, having served in that capacity for the past seven years.

During the day, one foundry reported that graphite produced higher quality steel than did bituminous

coke. They produced steel finishing as low as 0.018 pct sulphur in heats made with all steel scrap and no pig iron or cast iron scrap in the charge.

It was also reported that a new form of carbon, now called gas coke, produced from natural gas, and 99.5 pct pure carbon, will be tested soon as a pig iron substitute.

Specifications of raw materials such as lime and fluorspar particularly in cold metal practice was the second major item on the program. Proper sizing of fluorspar and the possibility of avoiding lime, which is overburned, was stressed in addition to ideal chemical specifications. A rapid, effective test of the degree of burn for lime was described as a fracture test which would show a small core of metal that was only slightly burned.

In addition to the use of lime and fluorspar for sulphur reduction, one shop explained how manganese ore and spiegel were effectively used to combat sulphur. When firing with oil, besides increasing the amount of pig iron in the charge by 2 pct, manganese ore is charged on top of the lime to increase the residual manganese at melt down. Further addition of manganese in the form of spiegel is made as soon as the heat "opens up" under a good slag while there is plenty of action in the bath. This addition of specially sized 1 in. to 2 in. crushed spiegel, which melts rather rapidly, allows a more rapid absorption of sulphur at the slag metal interface. Sulphurs under 0.020 pct are consistently maintained with this practice.

Experience on the use of basic ends on open hearth furnaces in the Southwestern Section indicates that its greatest advantage is the elimination of considerable down time for repairs. Discussion regarding the use of block or burner arch indicated that some plants showed unfavorable results when the block arch was removed.

The only plant in the area using oxygen extensively to reduce carbon and increase production reported that October production was 14.50 tons per hr for the shop. This compares with a shop average of 12.75 tons per hr the previous month when no oxygen was used. Although the use of oxygen does reduce the open hearth furnace roof life slightly, overall cost is not substantially increased because of a fuel saving of approximately 225,000 Btu per ton of ingots produced.

Of special interest to the basic steel foundries was a paper on the use of bath temperature control and its effect on quality. It was shown that a narrow range of tapping temperatures is required for first grades of steel, to eliminate casting defects. When tapping too hot, "sand burns" or dirty steel results; when tapping too cold, "cold shuts" are caused. Based on experience, there no longer seems to be any doubt that equipment now available to the melter makes it possible to control tapping temperatures within close range.

Connecticut Section

John L. Christie, metallurgical manager of Handy & Harmon, provided the Mar. 8 meeting of the Connecticut Section with a comprehensive survey of the production, uses, and properties of silver and its more widely known alloys. L. E. Thelin, section chairman, did the welcoming honors. Special recognition was given the eight students present in the form of a technical book, presented by W. R. Hibbard who introduced each of the students. Total attendance was 39.

Mr. Christie compared the physical properties of silver with other metals, mentioning its relatively high density and low modulus of elasticity, and discussed the alloy systems, of which silver-copper is perhaps the most important.

During the afternoon of Mar. 8, preceding the evening technical session, 27 members and guests visited the strip mill of the Detroit Steel Corp., Reliance Steel Division, at Hamden, Conn.

1950 Gordon Research Conferences

The Gordon Research Conferences for 1950, sponsored by the American Association for the Advancement of Science, will be held from July 3 to Sept. 1 at the New Hampton School, New Hampton, N. H. The conferences were established to stimulate research in universities, research foundations, and industrial laboratories. The purpose of the program is not to review the known fields of science, but to bring experts up-to-date on late developments, analyze the significance of developments, and provoke suggestions as to underlying theories and profitable methods of approach for further progress.

The first session, to be held from July 3 to July 7, will be on the chemistry and physics of metals, with Earl A. Gulbransen acting as chairman. Morris Cohen represents the AIME, Institute of Metals Div., which is cooperating in planning this program.

Information concerning the conference may be obtained from W. George Parks, director, Dept. of Chemistry, Rhode Island State College, Kingston, R. I. Applicants must state company or institution affiliation and the work in which they are most interested.

Program of the Gordon Research Conference, Chemistry and Physics of Metals

The Metallic Bond

July 3

"Atom Bombs in Metals," by S. Dushman.

"Solid State Theory and Metallic Binding." Speaker to be announced.

"Some Theories of Intermetallic Chemistry," by D. Harker.

July 4

"The Crystal Structure of Some Intermetallic Compounds," by D. P. Shoemaker.

"Bonding in Semi-metals," by L. Brewer.

"Bonding in Interstitial Compounds," by R. E. Rundle.

July 5

"The Relationship between Measurable Thermodynamic Properties and Theories of the Solid State," by C. Wagner.

"Experimental Methods for Evaluating Cohesive Energies, Free Energies and Residual Entropies of Simple Alloys as Illustrated by the Mg-Cd. System," by E. Wallace.

"The Distribution and State of Hydrogen in Metals and Certain Inferences," by D. P. Smith—presented by G. Derge.

"Neutron Diffraction Studies in Relation to Solid Structure," by C. G. Shull.

July 6

"Metallic Binding near Surfaces and Lattice Defects," by H. B. Huntington.

"Relationships between Elastic Constants and the Effect of Pressure and Metallic Binding." Speaker to be announced.

"A Survey of X-Ray Studies on the Electronic Band Structure of Metals," by C. H. Shaw.

July 7

"Theoretical Predictions of Binary Phase Diagrams," by R. Speiser and J. W. Spretnak.

Personals

Charles M. Kay has been appointed division superintendent of steel works of the American Steel & Wire Company's South Works in Worcester, Mass. He has been with the company since 1936, as observer and works metallurgist at Allentown, division metallurgist at South Works, in the Cleveland office, and as assistant division superintendent at South Works.

Willis S. Cady graduated from the Missouri School of Mines in January and has gone to work for the American Car & Foundry Co. in the St. Louis area.

S. F. Carter, assistant melting superintendent of the American Cast Iron Pipe Co., Birmingham, Ala., received the annual Steel Foundry Facts prize award for excellence of material published in the Steel Founders' Society's technical periodical, at the Society's annual meeting in Chicago in March.

A. O. Dufresne, deputy minister of the Department of Mines, Quebec, has been elected president of the Canadian Institute of Mining and Metallurgy for 1950-51.

Edward P. Fleming, who has been a consulting metallurgist with the American Smelting and Refining Co. since 1929, retired on Feb. 1, but will continue with the company as a part-time consultant. Mr. Fleming went to work for AS&R 44 years ago at the Garfield smelter and later visited both U. S. and South American properties. He went to the company's New York office in 1929 as a consulting metallurgist, returning to Salt Lake City in 1933.

Hugo L. Johnson has been manager of the Midvale plant of the U. S. Smelting Refining and Mining Co. since the first of the year. A year ago this month he was made general superintendent at Midvale and since 1933 has been closely allied with Midvale affairs.

Robert E. Savage

Robert E. Savage recently joined the magnesium - containing cast iron section of the development and research division of the International Nickel Co. in New York to work on the company's new ductile iron. Mr. Savage has been associated with Bethlehem Steel, Carnegie - Illinois Steel, and the Pittsburgh Steel Foundry.

Sidney Rolle, assistant manager of the Scomet Engineering Co., left in April for England and the Continent where he will visit metallurgical plants interested in large capacity induction furnaces for melting nonferrous metals.

Joseph W. Joyce is working for the Vermont Copper Co. at South Strafford, Vt. as metallurgist.

COMING EVENTS

MAY

1 **Boston Section, AIME.**
1 **Pittsburgh Section, AIME, Mellon Inst. Auditorium.**
2 **Washington, D. C., Section, AIME.**
2 Soc. of Applied Spectroscopy, New York.
5 **Columbia Section, AIME.**
6 **Chicago Section, AIME, Hotel Shoreland, Chicago, dinner-dance honoring Pres. McLaughlin.**
10 **El Paso Metals Section.**
10 **San Francisco Section, AIME.**
12 **Rio de Janeiro Section, AIME.**
15 **Detroit Section, AIME.**
17 **Executive and Finance Committee, New York.**
18-19 **Galvanizers Committee, sponsored by Amer. Zinc Inst., spring meeting, Toronto, Ont.**
24 **Iron & Steel Div., AIME, annual May dinner, Engineers Club, New York.**
24-25 American Iron and Steel Institute, Waldorf-Astoria Hotel, New York City.
28 Soc. for Applied Spectroscopy, 5th annual meeting, New York.

JUNE

6-17 Second Mechanical Handling exhibition and convention, Olympia, London.
11-15 Amer. Electroplaters' Soc., 37th annual convention, Hotel Statler, Boston, Mass.
16-17 **Central Appalachian Section and Coal Division, AIME, Daniel Boone Hotel, Charleston, W. Va.**
19-23 American Soc. for Engineering Education, Seattle, Wash.
26-30 ASTM 53rd annual meeting and 9th exhibit of testing apparatus and equipment, Atlantic City, N. J.

JULY

3-7 Gordon Research Conferences, AAAS, on chemistry and physics of metals, New Hampton School, New Hampton, N. H.
10-15 World Power Conference, London.
30-31 Mining Association of Montana, Hotel Finlen, Butte.

SEPTEMBER

5-9 6th National Chemical Exposition, Chicago Coliseum, Chicago.
15-16 National Soc. of Professional Engineers, Wheeling, W. Va.
18-23 3rd International Mechanical Engineering Congress, Brussels.
19-23 Conference on Wire Ropes in Mines, Ashorne Hill, Warwickshire, England.
26-29 Assoc. of Iron and Steel Engineers and Iron and Steel Exposition, Cleveland Public Auditorium, Cleveland.

OCTOBER

4-5 Texas Mid-Continent Oil & Gas Assn., annual meeting, Dallas.
9-13 **Washington, D.C., Section & Mineral Economics Division, AIME, joint meeting, Shoreham, Washington.**
13 **Southwestern Section, Open Hearth Committee, Iron and Steel Division, Houston, Texas.**
20 **Eastern Section, Open Hearth Committee, Iron and Steel Division, annual all-day fall meeting, Warwick Hotel, Philadelphia.**
23-27 National Metal Congress, Chicago.
23-25 **Institute of Metals Division, AIME, fall meeting, Hotel Sheraton, Chicago.**
23-27 ASM, Chicago.
23-27 American Welding Society, Chicago.
23-27 Soc. for Nondestructive Testing, Chicago.
27 **Southern Ohio Section, Open Hearth Committee, Iron and Steel Division, annual all-day meeting, Deshler-Wallick Hotel, Columbus.**

NOVEMBER

3 **Pittsburgh Section, Open Hearth Committee, Iron and Steel Division and Pittsburgh Section, AIME, annual all-day meeting, William Penn Hotel, Pittsburgh.**
9 AMC, coal division conference, William Penn Hotel, Pittsburgh.
16-18 GSA, annual meeting, Hotel Statler, Washington, D. C.

DECEMBER

7-9 **Electric Furnace Steel Conference, Iron and Steel Division, Hotel William Penn, Pittsburgh.**

FEBRUARY 1951

19-22 **AIME, annual meeting, Jefferson Hotel, St. Louis.**

APRIL 1951

2-4 **Open Hearth and Blast Furnace, Coke Ovens and Raw Materials Conference, Iron and Steel Division, Statler Hotel, Cleveland.**

Alvin Herzig, who has been chief metallurgist of the Climax Molybdenum Co., of Illinois since 1931, has been made president.

Robert E. Powers is a fellow at the Mellon Institute of Industrial Research, University of Pittsburgh. He had been with the Koppers Co.

Robert N. Honeyman is working with the Keystone Carbon Co., St. Marys, Pa., in powder metallurgy. His home is at 454 Parade St., St. Marys.

Ernest G. Kendall, formerly an instructor of metallurgical engineering at the Polytechnic Institute of Brooklyn, is now with the development department of the Titanium Division of the National Lead Co. at Sayreville, N. J.

Carl A. Liedholm is a consulting engineer with the Mesco Engineering Service, metallurgical and welding engineering services, West Englewood, N. J.

William R. Opie is working as research metallurgist for the Titanium Division of the National Lead Co., South Amboy, N. J.

Edmund D. Haigler

Edmund D. Haigler has joined the Fischer & Porter Co. in Hatboro, Pa., to work on special customer engineering problems. Since graduating from Harvard, he has stored up twenty years of experience in industrial instrumentation with the Chevrolet Division of General Motors, E. I. du Pont de Nemours & Co. on ammonia, cellophane, and rayon, and most recently with the Foxboro Co. He managed Foxboro's engineering department for several years, and recently completed a two-year survey of the mining, milling, and smelting industries.

Thomas E. Lloyd

S. F. Ravitz has left the Salt Lake City station of the Bureau of Mines to join the extractive metallurgy department at the University of California at Berkeley.

Thomas E. Lloyd has been appointed manager of publications of the AIME, succeeding **T. W. Lippert** who resigned recently to accept the position of general manager of the newly-formed Titanium Metals Corp. of America, owned jointly by Allegheny Ludlum Steel Corp. and National Lead Co.

For the past 10 years, Mr. Lloyd has held various editorial positions with **The Iron Age** in Detroit, Cleveland, and Pittsburgh, and in 1948 was appointed machinery editor. Prior to joining **The Iron Age**, he had been with the Jones & Laughlin Steel Corp., Pittsburgh.

Francis M. Rich, vice-president of the Kaiser Steel Corp. in charge of operations at Fontana, Calif., has become general superintendent of the Indiana Harbor Works of the Inland Steel Co. Mr. Rich has worked for Carnegie-Illinois, Wisconsin Steel, Youngstown Sheet & Tube, Republic Steel, and the Steel Co. of Canada, Hamilton, Ont.

Clifford E. Wenninger resigned as chief metallurgist of the Crawford Steel Foundry Co. to become foundry engineer in the department of mining and metallurgy at the University of Kentucky. He is working on steel foundry research problems sponsored by different private firms and the Steel Founders' Society of America.

Philip R. White, after receiving his M.S. degree in metallurgy from Columbia in October, started work in the metallurgy department of the General Motors research laboratories in Detroit.

Obituaries

H. W. Gillett

AN APPRECIATION BY J. G. THOMPSON

H. W. Gillett died from a heart attack on March 3, near Lexington, Ky., on his way home to Columbus after an extended visit to Georgia and Florida.

Horace Wadsworth Gillett was born Dec. 12, 1883 in Steuben County, N. Y. He received an A.B. from Cornell in 1906 and subsequently worked as a chemist for Thomas A. Edison and for A. D. Little, returning to Cornell in 1908 for his Ph.D. He was manager of the research department of the Aluminum Castings Company, 1910 to 1912; chief alloy chemist of the USBM assigned to the Ithaca, N. Y., station, 1912 to 1924; chief of the metallurgy division of the National Bureau of Standards, 1924 to 1929. Since 1929 he was associated with Battelle Memorial Institute, as its first director, 1929 to 1935, as chief technical advisor from 1935 to his retirement at the end of 1948, and subsequently as consultant.

He became a member of the AIME in 1918 and was Howe Lecturer in 1939. He was Edwin Williams

NECROLOGY

Date Elected	Name	Date of Death
1943	Tom Lee Ball	June 24, 1949
1934	John B. De Mille	Unknown
1935	Robert L. Klotz	Unknown
1899	Henry M. Lane	Aug. 8, 1945
1914	George D. Nordenholt	May 23, 1949
1915	Thomas C. Peddar	Oct. 27, 1949
1942	Herbert T. Quin	June 14, 1949
1945	P. C. Robbins	Feb. 19, 1950
1916	P. M. Saxman	Apr. 27, 1949
1897	W. W. Taylor	Unknown
1937	Arnulfo Vega	May 15, 1949
1938	Nicolas N. Vishnevsky	Mar. 19, 1950
1927	Richard A. Wagstaff	Mar. 8, 1950
1916	William A. Wasley	Feb. 7, 1950

Lecturer for the Institute of British Foundrymen in 1942, and delivered the Second Annual Foundation Lecture of the American Foundrymen's Association in 1944. He received the McFadden Medal of the American Foundrymen's Association in 1932 and was editorial director of "Metals and Alloys", 1929 to 1942. He was a member of the War Metallurgy Committee of the Office of Scientific Research and Development.

His activities covered the entire field of ferrous and nonferrous metallurgy. In his 12 years with the Bureau of Mines, he continued his early work on aluminum and its alloys, developed the rocking type of indirect arc furnace for melting brass and other metals, and directed the Bureau of Mines World War I investigation of special steels, including uranium steel. This investigation was one of the first to call attention to the merits of molybdenum as an alloying element in steel. At the National Bureau of Standards and subsequently at Battelle, his influence was evident in all of the varied metallurgical activities which he directed.

His ability to assemble and correlate vast quantities of scattered, disconnected facts and data into a compact, clear, and concise whole was a constant wonder to his associates. He was truly a leader, who led by example and never drove anyone but himself. In the laboratory or on his occasional outings for relaxation, if there was ever an unpleasant or disagreeable job to be done his associates usually became aware of the job when they found that it was already under way in Gil's hands.

He will be remembered as a leader in the physical metallurgy of his era, not only for his personal contributions but for his ability to stimulate his associates to better work than even they had recognized that it was possible for them to do.

Arnulfo Vega (Member 1937), manager of Cia. Minera Idria, Concheño, Chih., Mexico, died May 15, 1949. Mr. Vega was born at Meogui, Chih., in 1916, but went to El Paso when a child, and earned his metallurgy degree at Texas Western. He went to work for Minas de Pinos Altos as a millman, eventually becoming mill superintendent. When that company was liquidated in 1946 he went to Concheño to supervise the installation of the new flotation mill of Cia. Minera Idria. He was subsequently named general superintendent and finally manager.

Selden Scott Rodgers (Member 1913), superintendent of concentrators at the Anaconda reduction department of the Anaconda Copper Mining Co., died on Jan. 8. Born at Toledo, Ohio, in 1886, and educated at Harvard, he went to Great Falls, Mont., in 1910 and worked for Anaconda since then, with the exception of the summer of 1912 which was spent in Alaska. He was president of the Quartz Hill Mining Co. and an active member of the Montana Mining Association.

Robert Ray Van Valkenburgh (Member 1915), eastern states manager for the Globe Union Mfg. Co., Philadelphia, died Oct. 29, 1949. After completing his studies at the Michigan College of Mines, Mr. Van Valkenburgh spent several years in Alaska with the Alaska Gastineau Gold Mines. Back in the States again he worked for the Triangle Mining Co., the Utah Cons. Mines Co., the Wharton Steel Co., and the Empire Steel and Iron Co.

Ernest Waters Wescott (Member 1927), of Comstock & Wescott Inc., Niagara Falls, N. Y., died Jan. 14. Born at West Newton, Mass., in 1888, and educated at Harvard and MIT, he worked as assistant professor of chemical engineering at MIT in charge of the Niagara Falls Station and then went on to the Nitrogen Products Corp. He managed Metals Chemical, Ltd., Welland, Ont., for a couple of years and then was put in charge of the Niagara Falls branch of the company which was the predecessor of Comstock & Wescott. Mr. Wescott was also president of Metal Development Corp. and a director of Longyear Process Inc.

Proposed for Membership

Total AIME membership on Mar. 31, 1950, was 16,397; in addition 4037 Student Associates were enrolled.

ADMISSIONS COMMITTEE

E. C. Meagher, Chairman; Albert J. Phillips, Vice-Chairman; George R. Corless, Lloyd C. Gibson, Ivan A. Given, P. Malozemoff, Richard D. Mollison, and John Sherman.

Institute members are urged to review this list as soon as the issue is received and immediately to wire the Secretary's office, night message collect, if objection is offered to the admission of any applicant. Details of the objection should follow by mail. The Institute desires to extend its privileges to every person to whom it can be of service but does not desire to admit persons unless they are qualified.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; AM, Associate Member; S, Student Associate.

Arizona

Ajo—Young, Joe Bay. (C/S—S-J).
Clarkdale—O'Brien, Richard Clarke. (C/S—S-M).
Ray—Barkley, James Robert. (C/S—S-J). Williams, Robert I. (C/S—S-J).
Safford—Tomkinson, Joe Alfred. (M).
Superior—Pillar, Charles Littlefield. (R,C/S—J-M).

California

Bakersfield—Davis, Chester Arthur. (C/S—J-M). Marshall, Warren Monroe, III. (J).
Glendale—Parker, Richard Lee. (C/S—S-J).
Huntington Park—Farrow, John Robert. (J).
Los Gatos—Frye, Everett Stanton. (C/S—AM-M).
Los Angeles—Barr, Gordon William. (R—M). Fernandes, Robert John. (C/S—S-J). Sampson, Norman Nathan. (C/S—J-M).
McKittrick—Campbell, Harry Duncan. (C/S—AM-M).
Monterey Park—Harber, Delmont Davis. (M).
Ontario—Greene, William Ambrose. (C/S—S-J).
Pasadena—Cerf, Charles Joseph. (M).
Richmond—Pratt, William Held. (C/S—J-M).
San Francisco—Reno, Robert Evans, Jr. (M).

Colorado

Climax—Walker, Marvin Stanley. (R,C/S—J-M).
Colorado Springs—Magruder, James Alexander. (AM).
Denver—Newton, Douglas Edmund. (C/S—S-M).
Pueblo—Gabelman, John Warren. (C/S—S-J).
Uravan—Clark, Henry D. (M). George, Jack Wesley. (R,C/S—J-M). Tschanz, Charles McFarland. (C/S—S-J). Van Fleet, Thomas Ricard. (R,C/S—S-J).

Connecticut

Darien—Griffin, Frank Alexander. (M).
Fairfield—Marston, William Thomas. (M).
New Canaan—Marshall, Carl Alexander. (M).
Southington—Francis, Leonard William. (R,C/S—S-AM).
Stamford—Hartjens, Hermen. (C/S—J-M).

Delaware

Wilmington—Desy, Donald Herbert. (C/S—S-J).

Florida

Lakeland—Linder, Paul Scott. (J).

Illinois

Chicago—Clark, Horace Herbert. (R—M). Rumfelt, Henry

F. C. (M). Senior, Franklin Cumming. (M).
DePue—Wyman, George Francis. (M).
LaGrange—Burrack, Harry Bob. (M).
Maywood—Koebel, Norbert Kenneth. (M).
Park Forest—Hall, Lewis Dana. (J).
West Frankfort—Karlovsky, Jerry, Jr. (C/S—S-J).

Indiana

Hammond—Bradley, Robert Calvin. (C/S—S-J).
Terre Haute—Lentz, Jerome Victor. (AM).

Kansas

Eureka—Campbell, William Henry. (J).

Kentucky

Ashland—Miller, Robert Jackson. (M). Todd, George Hensel. (M).

Louisiana

Monroe—Ford, Elbert Estelle. (M).

Maryland

Silver Spring—Spendlove, Max Jones. (AM).

Michigan

East Lansing—Sweet, Robert Louis. (C/S—AM-M).
Houghton—Goodyear, Frank Taylor Crim. (C/S—S-J).

Minnesota

Crosby—Stetson, Harland John. (C/S—S-J).

Mississippi

Laurel—Hamilton, Walter Frederick. (C/S—J-M).

Missouri

Joplin—Austin Arthur Bradford. (AM).

Montana

Mike Horse—Augustson, Anders Everet. (C/S—S-J).

Nevada

Boulder City—Baker, Don. H., Jr. (C/S—S-J).
Ruth—Brown, Arnold Lee. (C/S—S-J).

New Jersey

Chatham—Pfann, William Gardner. (C/S—J-M).
East Paterson—Franklin, Frederick Francis. (M).
Hohokus—Burrows, Henry Peter. (AM).
Westfield—Mayne, Charles Russell. (M).
West Orange—Driver, Robert Orrell. (M).

New Mexico

Carlsbad—Clark, Harold P. (R,C/S—S-M).
Silver City—Davis, Richard R. (AM).

New York

Bronxville—Pels, Alan Robert. (J).
Forest Hills—Speaker, Francis Braun. (C/S—AM-M).
Katonah—Davis, Donald Wilfred. (C/S—J-M).
Newton Falls—Ross, Alexander, Murray. (M).
New York—Piana, Louis D. (C/S—S-J).
Troy—Deshpande, Raghunath Chintaman. (C/S—S-J).

Ohio

Ashtabula—Yake, William Harry. (C/S—S-J).
Bellaire—Jordan, Luis, Jr. (C/S—J-M).
Blacklick—Newton, William Harding. (R—M).
Cincinnati—Kahles, John Frank. (C/S—J-M). McConnell, William Cameron. (M).
Cleveland—Bradley, Morris Alva. (M). Robards, Chester Frazier. (R,C/S—S-M). Zorko, Frank Anton Valentine. (C/S—S-J).
Lakewood—Morrison, Donald M. (C/S—AM-M).
Youngstown—Gayton, Oscar Francis. (M).

Oklahoma

Bartlesville—Tarner, Jack. (C/S—J-M).
Duncan—Pendley, Dee Finas, Jr. (J).
Enid—Duff, Thomas H. (C/S—S-J).
Norman—Porter, Alfred Watson, Jr. (J).
Oklahoma City—Durham, Elbert Norvin. (R, C/S—J-M). Hill, Freeman Lee. (C/S—S-J).
Tulsa—Elliston, Henry Harold. (M). Johnson, Douglas Ostrom. (R—M). Knotts, Max Leon. (C/S—S-J). Muratta, Robert Wyman. (AM). Stafford, Joseph Dewey, Jr. (J). Willson, Charles Otis. (M).

Oregon

Corvallis—Robberson, Ronald Frank. (C/S—S-J).
Portland—Cappa, Francis Xavier. (C/S—J-M).

Pennsylvania

Bridgeville—Marquis, Bernard Ernst. (M).
Carbondale—Colville, Kenneth Hendrick, Jr. (J). Dann, Benjamin George, Jr. (J).
Easton—Miltenberger, Robert Stanley. (J).
Finleyville—Blair, James Philip. (J).
Hazleton—Mitchell, Donald Leroy. (AM).
Lansford—Turrall, William Thomas. (M).
Mount Carmel—Rothrock, Howard Eugene. (M).
Nanticoke—Jones, Donald William. (M).
Palmerton—Taylor, Charles Edward. (M).
Philadelphia—Rassbach, Herman Philip. (M).
Pittsburgh—Bauman, Arthur Harvey. (C/S—S-J). Fulton, Paul Franklin. (C/S—J-M). Gorman, Henry Benjamin. (J). Shaw, James Marshall. (J). Wyllie, Malcolm Robert Jesse. (M).
State College—Mullen, Willard F. (M).

South Dakota

Estelline—Burge, Furman H., Jr. (C/S—S-J).

Texas

Beaumont—Cokinos, Geneos Pete. (R, C/S—S-M).
Bellaire—Kent, William Dexter. (R,C/S—J-M). Morris, Wesley Dixon. (C/S—J-M). Sherwin, Mel William. (M).
Brownfield—Ray, Jack Harris. (C/S—S-J).
Corpus Christi—Wallace, Robert Roy. (C/S—S-J).
Dallas—Monkhouse, George Sedberry. (C/S—J-M).
El Paso—Foester, Hallard Lewis. (AM).
Gladewater—Zlomke, Charles Donald. (J).
Hawkins—Kempe, Gerald Lloyd. (J).
Hebbronville—Thomas, Henry Evans. (J).
Houston—Blouin, Cecil Francis. (M). Cain, Lester Lee. (J). Cook, Winfield Hollis. (M). Floeter, John Sydney. (R,C/S—S-M). Nice, Frank Allen, Jr. (M). Wright, Randolph Earle. (J).
Midland—Brown, Billie Joe. (J). Brown, Jack Weldon. (J). Clark, Joe Richard. (J). Gillett, Dan. (J). Guenther, Richard Stewart, Jr. (C/S—S-J). Keeler, William Arnold. (J). Landtiser, Paul Keith. (J). Tompson, Gilbert Carr. (J). Trott, James Irby. (R,C/S—S-J).
Odessa—Qusterhout, Raymond Sherburn. (M).
Pleasanton—Kirby, John Elmer, Jr. (R,C/S—S-J).
Scroggins—Grolemund, John Lafayette. (J).
Seagraves—Hutchins, Charles H. (C/S—J-M).
Texas City—Naylor, Roger A. (J).
Wink—Purdy, Eugene William. (J).

Utah

Orem—Hinds, Gene Wesley. (C/S—S-J).
Salt Lake City—Hodgins, H. B. (M).
Spring Canyon—Pauley, Clarence Everett. (M).

Vermont

South Strafford—Joyce, Joseph William. (C/S—S-J).

Washington

Everett—Irvine, Gordon Benjamin, Jr. (C/S—S-J).
Spokane—Taber, John William. (C/S—J-M).

West Virginia

Beckley—Wilson, Crawford Lindsay. (M).
Huntington—Smith, Herman Eugene. (J).
Montcoal—Kane, Chester Opal. (M).

Wisconsin

Marinette—Lindlof, Raymond George. (C/S—S-AM).
Shullsburg—Lasio, John. (R,C/S—S-M).

Wyoming

Riverton—Johnston, Myron Thomas. (M).
Thermopolis—Allen, Thomas Newton. (M).

Alaska

Anchorage—Harlan, Garth Byron. (C/S—S-AM). Robinson, Wiley David. (M).

British Columbia

North Vancouver—Hefferman, Gerald Robert. (M).

Ontario

Welland—Preston, Thomas B. (C/S—S-J).

Quebec

Abitibi—Dumont, Georges H. (M).

Cuba

Habana—Fisher, Arnold Joseph. (M).

Trinidad

San Fernando—Holte, Justin LeRoy. (C/S—S-J).

Argentina

Jujuy—Guerrico, Martin. (J).

Brazil

Sao Paulo—Correa Da Silva, Luiz Coelho. (C/S—S-J).

Chile

Chuquicamata—Cadwell, Graham Matthews. (M).

Ecuador

Guayaquil—Avery, Don. (C/S—S-J).

Venezuela

Caracas—Sykes, Harry John. (C/S—J-M).

France

Meuse—Arnould, Andre Marcel. (M).
Paris—Armanet, Jean Charles. (R—M).

Gold Coast Colony

Tarkwa—McMillan, Alexander Marshall. (M).

India

Jaipur City—Sethi, Munshi Lal. (C/S—S-J).
Madras—Narasimhan, Asthagiri Subbaiyer. (J).

Japan

Kagawaken—Yamagath, Shiro. (M).

New South Wales

Sydney—Hamilton, Frank Alexander. (C/S—J-AM). Richardson, Gilbert Robert. (C/S—J-M).

New Zealand

Wanganui—Billinghurst, William McMinn. (C/S—S-J).

Effect of Iron Quality On Bessemer Blowing Problems

by R. E. Edwards

PRIOR to World War II, progress of the acid Bessemer process reached a standstill, even though the use of Bessemer steel had been extended to the manufacture of seamless pipe and for dephosphorization in routine production. The duplex openhearth process had been developed to a point that with certain practice restrictions, a product with physical properties equivalent to openhearth scrap practice steel could be produced. However, metallurgical requirements imposed by the trade became increasingly restrictive and many applications previously open to Bessemer were changed to prohibit its use. A continuation of the trend prevailing at the time undoubtedly would have resulted in a decrease in Bessemer ingot production.

World War II, however, changed this picture. With unprecedented demands for steel products of all types, the Bessemer converter, a most rapid steel producing method, proved valuable, whether used to convert Bessemer iron into Bessemer ingots or to prepare basic iron for further refinement in the basic openhearth furnace. Demand for steel was accompanied by a similar demand for steel melting scrap, an absolute requirement where facilities did not include the Bessemer converter. A fairly uniform relationship between the price of steel melting scrap and duplex openhearth ingot production explains, in part, the fluctuations in duplex production. A drastic rise in scrap prices ruined this relationship and may partly explain the converter plant's present popularity. This increased interest resulted in the expansion and erection of new converter facilities.

The raw material supply changes have caused concern, and their effects have been felt. Geologists, raw materials divisions, planning boards, and blast furnace operators have been most affected. However, the Bessemer converter operator ultimately may be affected to the greatest degree. Practices established through years of operation are not easily altered, and necessary raw material changes will be acceptable only if accompanied by changes in practices in both blast furnace and converter operations.

These changes are:

(1) Depletion of metallurgical quality coal. This will eventually result in a poorer quality coke, both physically and chemically. The connection between coal and coke quality and converter operations is evident in the blast furnace man's explanation for extreme variations in silicon and sulphur and inferior iron quality. This answer is usually "poor coke."

(2) Depletion of low phosphorus, hematite, Bessemer lake ores. In some instances this has resulted in the substitution of lower iron content, magnitite ores that require beneficiation and sintering treatments to be suitable for blast furnace smelting.

(3) Depletion of low manganese ores, principally of the basic type. This will result in the use of higher manganese ores and iron with a manganese content higher than heretofore considered desirable in blowing iron.

With the iron handling practice best suited to each individual plant, iron of an analysis and physical temperature that falls within a controllable range is delivered to the converter. Bessemer iron as free as possible of nonmetallic materials, is desirable. After charging to the converter, the blow is expected to be completed with a minimum loss of the charge by ejection, since the problem of cleaning up ejected material is almost as serious as yield loss. The quality and amount of this material also determines the extent to which yield loss can be recovered by recharging it as a coolant.

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Clean separation of slag and metal during emptying of the converter is important to minimize nonmetallic contamination of the steel, maintain a uniform efficiency of ferroalloy additions, and facilitate removal and subsequent disposal of the slag product. Scheduled operation also depends on a uniform life of converter linings, which may be adversely affected by excessive erosion or building up of the lining, both of which occur.

There are converting mill problems thought to be most closely associated with iron quality, which in turn will be affected by the raw material changes described. These problems are temperature control, slopping and lining life. Because of differences in mechanical layout, operating practices, raw materials, and type of product, converter operating problems vary between individual plants. Disagreement may exist as to the origin of various problems and the most practical methods for correcting them.

In the problem of temperature control of metal in the Bessemer converter, silicon contained in the iron is the most important single factor, since its oxidation is the major heat source. The importance of silicon content and the extent to which permissible ranges are affected by other operating conditions is illustrated by the wide variety of specifications set up in various converter plants.

These differences are a result of different methods of handling iron between the blast furnaces and the converting mill, methods of controlling the temperature of the blow in the converter, and the temperature desired in the blown metal product.

Since iron analyses are not always available at the time of blowing, particularly with the faster methods of iron handling, unpredictable analysis changes in the production of a furnace from cast to cast are burdensome to the Bessemer operator. Variations may also be expected within a cast. One instance is illustrated in table I. Silicon content within the cast varies about 32 pct, from 1.19 pct to 1.51 pct. This variation has been evened somewhat by casting into submarine mixer ladles, but the blower must still compensate for a 0.15 pct variation. This is important since it is one-half of the specified range, in this case, from 1.00 pct to 1.30 pct.

Also important to the blower is the temperature of the iron to be charged. High temperature at charging reflects high temperature at cast and

TABLE I

Iron Analysis Variations, No. 3 Furnace, Basic Iron

	Analyses			
	Si	P	S	Mn
CAST SAMPLES:				
At First of Cast	1.29	0.140	0.039	0.58
At ¼ of Cast	1.51	0.142	0.022	0.63
At ½ of Cast	1.39	0.138	0.027	0.65
At ¾ of Cast	1.19	0.140	0.034	0.58
Average	1.34	0.140	0.031	0.61
SUBMARINE LADLE SAMPLES, IRON WITHDRAWN DIRECT:				
First Ladle Cast, 350,000 Lb				
Beginning	1.25	0.148	0.032	0.60
Middle	1.23	0.148	0.027	0.61
End	1.27	0.142	0.033	0.60
Second Ladle Cast, 147,000 Lb				
Beginning	1.38	0.116	0.027	0.53
Middle	1.36	0.128	0.025	0.53
End	1.36	0.116	0.027	0.54
Average	1.31	0.133	0.029	0.57

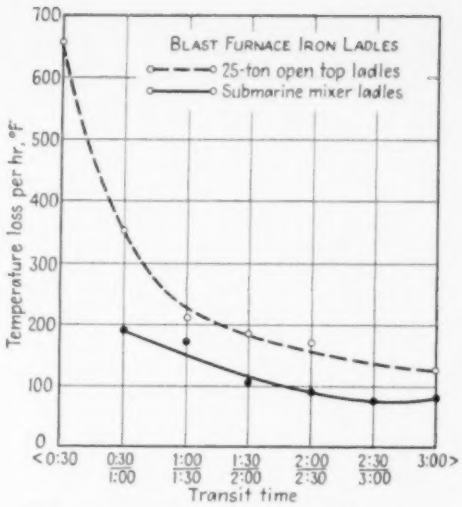

Fig. 1—The effect of temperature of iron to be charged into the Bessemer. High temperature at charging is associated with iron cleanliness, with variations depending on handling and time intervals.

is associated with iron cleanliness. Variations may be expected, however, depending on methods of handling and time intervals. The effect of these factors is illustrated in fig. 1. Data are based on temperatures determined by immersion thermocouples in 36 casts of Bessemer iron, with readings taken on 36 submarine ladles and 24 25-ton open top ladles, the two types of ladles being used concurrently. The initial temperature drop of 50° to 100°F from runner to ladle is included in the rate of temperature loss. The comparative loss in temperature between the two types of iron handling can easily be calculated.

The effects of extreme blown metal temperature variations are twofold; those related to extremely high temperatures and to very low temperatures. Excessive teeming temperature resulted in an almost complete absence of a protective metal layer between the ingot surface and subcutaneous blow holes, and poor surface will result regardless of ingot handling and heating practices. Excessive temperature also has prolonged solidification of the ingot enough to permit the formation of a pipe cavity, the source of laminations in product. The difference in nitrogen content between the two heats is also a function of temperature. The higher content in the hot heat will result in an inferior product with respect to cold forming characteristics. In duplex metal, the higher nitrogen content will either result in an inferior heat or require increased furnace time to reduce it to a normal value. Also excessive temperatures have an erosive effect on refractories such as openhearth furnace bottoms, ladle linings, stopper assemblies, and the vessel lining. Ingot mold and stool costs and the inconvenience and delay in handling sticker ingots also are adversely affected.

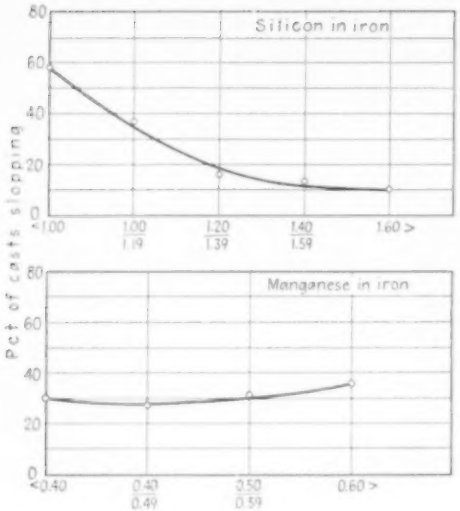

Fig. 2—The effects of silicon and manganese contents of the iron on slopping of the converter. The values used in plotting the curve are based on 2980 casts of basic iron, cast into and withdrawn from submarine ladles.

Typical of high blown metal temperature are higher than normal residual silicon contents, with values running as high as 0.20 pct compared to a normal 0.01 pct. Such values are intolerable in capped and keyed or rimmed Bessemer ingots. In duplex metal, where temperatures more often are high, a content of 0.20 pct in a normal duplex charge may increase the total silicon contained in the charge to a point that the normal flux cannot compensate for it, the lime-silica ratio of the slag is disrupted, and phosphorus reversion may result.

The effect of extremely low temperatures is obvious. In duplex practice, transfer ladles may be built up with frozen metal and metal charging hampered. The relation between skulled teeming ladles, bad pours, surface quality and metal cleanliness is well known. Metal loss through skulling also decreases yields.

Closer control over silicon content of blowing iron is the most desirable corrective measure for extreme temperature variations in Bessemer operation. Although having all analyses within the range specified as most suitable under prevailing practices would be ideal, low or high values probably could be handled if they were uniform. If variations could be anticipated, there are methods of correction. Scrap additions to the iron ladle, mixer and converter are used variously. This practice dilutes high silicon values to normal by adding low silicon scrap and consumers heat by the amount required to melt the addition.

These additions could be made more practical if extreme silicon variations could be anticipated, permitting scheduled additions to iron ladles before casting or if accurate temperature measurements were made at the cast or some stage of iron handling as a basis for additions to ladles or mixers. Coolant additions to the converter within the capacity of existing equipment would probably be adequate to supplement these controls. Steaming will continue until proved detrimental. Rapid analytical methods, such as the spectrograph, might permit desiliconization of iron known to be high in silicon content by an intermediate operation. Excess mixer or ladle capacity where iron is withdrawn directly, would permit the holding of iron to reduce physical temperature or to perform some intermediate operation such as desiliconization.

These methods might also help decrease the percentage of cold blows. Scrap additions would not be made to low silicon iron at any point, knowledge of analysis and physical temperature would be available to the blower before blowing, iron could be handled promptly, and methods designed to increase metal temperature during blowing could be applied.

The term slopping describes a variety of types of ejection of material from the mouth of the converter during blowing. The term is somewhat ambiguous, particularly with regard to the severity of the condition. At the beginning of a blow, fumes and any material not combined with the iron may be ejected by the sudden force of the blast. After the first stage of the blow and through the portion known as the silicon blow, the operation is relatively clean since the product of the oxidation of silicon is a solid and is retained in the converter. Under ideal conditions, the blow becomes clearer as the oxidation of carbon commences and the flame begins to lengthen as more carbon monoxide is burned at the mouth of the converter.

These ideal conditions do not always exist and, under certain circumstances to be described, the violent reactions associated with the evolution of great quantities of gas may result in a major explosion. The effect is spectacular to the uninitiated and discouraging to the operator, for as much as half of converter charge of metal and slag may be distributed around the landscape. This may be termed **middle of blow slopping** and if the blow is permitted to proceed, will cease as the rate of reaction decreases. The carbon blow, especially in its latter stages, is typified by the ejection of relatively small particles called **spittings.** These particles contain carbon (1.00 pct), iron oxide (33.00 pct), metallic iron, and small amounts of silica and manganous oxide. They solidify before landing and, being granular in form, do not present any serious problem. In certain instances, slopping similar to the middle of blow type may occur just prior to the completion of the blow. This addition, while not so violent, may continue as the converter is turned to a horizontal position and is referred to as end point or turndown slopping.

Factors that cause middle of blow and endpoint slopping, or loss of slag and metal from the converter during the blow, may be purely mechanical. When increased production is desirable, converters may be charged with amounts in excess of their rated capacity. This will increase the tendency to slop, since a greater volume will be contained in the same size vessel, and during

the most violent reactions occurring in the blow the contents may boil over.

Formation of accretions on the vessel lining serves also to reduce the volume of the converter, with a similar result.

Uniform blast pressure is essential to uniform converter operation. Unfortunately, nonuniformity does occur, particularly with steam driven turbines supplying blast to several operating units, and abrupt changes in pressure may increase the volume of air passed through the metal in the converter to the extent that some portion may be ejected. The operation of turning the converter down at the end of the blow results in some slopping normally, since increasingly large volumes of air are passed over the molten bath as more tuyeres become exposed. It is customary to reduce the blast at this time. Inability to reduce blast pressure will increase slopping at this point. Likewise, the rate at which the converter is urned down is important. Tendency to slop is also attributed to an improper ratio between total tuyere area (or blast volume) and unit volume of metal charged.

Assuming that mechanical features are proper, other factors cause slopping, these being chemical by nature and associated with the characteristics of the slag formed during the blow. This slag is composed of (1) the solid oxides of elements contained in the charge, mainly iron oxides, manganous oxide, titania and silica; (2) the product of the erosion of iron runners, ladle and converter linings; and (3) blast furnace slag carried over with the iron. The order of formation of the slag can be assumed.

Iron oxide is formed first to combine with the varying amounts of contaminating material present. As silicon oxidation proceeds, this slag is enriched with silica, later accompanied by manganous oxide. No great composition change occurs throughout the carbon blow until it nears completion. Then iron oxide content is increased slightly, and, after the endpoint, is increased abruptly during the afterblow.

Slopping occurs most frequently during the middle and at the end of the blow. The relation of slag consistency and slopping at these two danger points is not difficult to guess. Gaseous evolution during the violent reactions at the beginning of the carbon blow is readily passed through a solid lumpy slag, but is blanketed and inhibited by a fluid. A fluid slag may absorb the gaseous products and thus occupy a greater volume than intended. An increase in iron oxide at the end of the blow tends to thin out the slag, but not normally to the extent that slopping occurs. The presence of any compound that increases fluidity at this point furthers the trend and causes slopping.

The effect of various factors on slopping has been the subject of extensive investigations at Aliquippa Works in recent years. Studies of iron analysis, iron handling, observation and recording of blowing performance of all iron casts, sampling and analysis of hundreds of converter slags, and studies of the effect of slopping on the Bessemer and duplex product were involved.

Since silica contained in the converter slag is a major factor in determining its refractoriness, the percentage of silicon in the iron to be blown

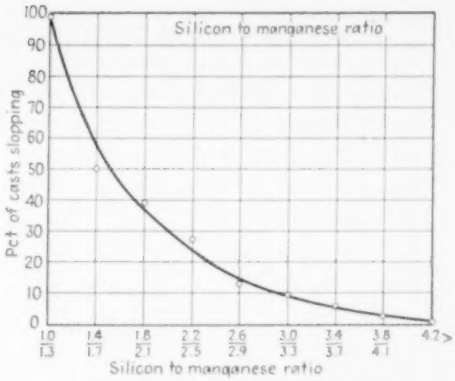

Fig. 3—The effect on slopping of the silicon to manganese ratio in the iron. The trend in this case is not distinct although a slight increase in slopping is indicated with higher manganese values.

is closely related to slopping. The normal slag contains at least 60 pct silica. Values below this amount are associated with a pasty or fluid slag, depending on other constituents, and higher values increase its fusion point. The effect of silicon content on slopping is shown in fig. 2. The values used in plotting the curve are based on the observation of the blowing performance of 2980 casts of basic iron, cast into and withdrawn from submarine ladles. The trend showing a decrease in slopping as silicon content increased is distinct. It should be noted, however, that slopping still occurred with silicon over 1.60 pct. This would indicate that silicon alone cannot control the problem.

High manganous oxide contents increase the fluidity of converter slags. The percentage contained is normally 8 to 10 pct with an extreme range from 6 to 14 pct, depending upon the manganese contained in the iron. The effect of manganese in the iron on slopping is also shown in fig. 2, using the same data as in the silicon curve, In this case, the trend is not distinct, although a slight increase in slopping is indicated with the higher manganese values.

Although silicon and manganese considered individually do not appear to be predominant factors in the occurrence of slopping, consideration of the two elements together presents an entirely different picture. The effect of the ratio of silicon to manganese on slopping is illustrated in fig. 3 by the same data as used in the silicon and manganese curves. The extreme range of values occurs, from slopping on all casts with ratios from 1.0:1.3 to a complete absence of slopping with values over 4.2. The reason for the establishment of minimum silicon to manganese ratios is evident. This minimum value varies, depending on other factors, but is usually set at 2.5, meaning that 0.50 pct manganese should be accompanied by at least 1.25 pct silicon.

The effect of titanium on blowing performance has been a matter of discussion, since magnetite ores, such as used in the production of sinter used

TABLE II

The Effect on Blowing Performance of Ferro-Titanium Additions to Iron

	Iron Analysis Si	Mn	Ti	Fe-Ti Addition to Iron Ladle	Blowing Condition	Converter Slag Analyses SiO_2	FeO	Fe_2O_3	MnO	CaO	MgO	Al_2O_3	TiO_2
A.													
1.	1.55	0.47	0.06	None	OK	64.04	16.13	2.57	10.90	0.45	0.23	1.74	2.36
2.				100 lb	OK	65.64	15.22	2.60	9.84	0.38	0.20	2.27	2.90
3.				150 lb	OK	61.50	20.25	1.00	8.71	0.34	0.28	2.09	4.46
4.	1.63	0.31	0.07	200 lb	OK	61.20	19.09	0.71	8.00	0.35	0.20	2.23	7.08
5.				None	OK	64.86	16.51	2.43	10.71	0.15	0.20	1.98	2.58
B.													
1.	1.18	0.44	0.05	None	OK	61.86	19.74	1.43	10.77	0.45	0.20	2.00	2.20
2.				600 lb	OK	57.34	20.64	1.00	9.88	0.49	0.35	2.32	6.86
3.				None	OK	62.82	16.77	2.30	10.38	0.57	0.22	2.03	3.50
C.													
1.	1.21	0.34	0.10	None	OK	64.24	21.05	0.57	10.12	0.29	0.20	1.63	2.18
2.				600 lb	OK	60.40	18.06	1.57	9.49	0.15	0.10	2.48	5.50
3.				None	OK	61.90	21.30	1.57	9.49	0.53	0.25	1.90	3.16

at Aliquippa Works, has comparatively high titania content. This material increases the titanium content of Bessemer iron from a normal 0.03 to 0.04 pct to values over 0.20 pct when 50 pct of the blast furnace burden is composed of magnetite sinter. The titania contained in the Bessemer slag blown from normal iron ranges from 1.00 to 3.00 pct, but is as high as 7.00 pct with 50 pct magnetite sinter in the burden. To achieve titanium contents existing when the burden contained high percentages of magnetite sinter, additions of 20 per cent ferro-titanium were made to clean blowing iron of 0.05 to 0.10 pct titanium content with 8 pct magnetite sinter in the burden. The effect of this experiment on slopping is shown in table II. In (A), successive additions increased the titania in the slag to 7.08 pct, compared to values of 2.36 pct and 2.58 pct on the preceding and following blows, without any apparent effect on blowing performance. In (B) and (C), larger additions of 600 lbs were not so efficient. Higher than normal titania contained in the slag did not affect blowing performance. A difference in the effect of residual and added titanium contents may exist. However, the results of these experiments have been verified by trials of high percentages of magnetite sinter producing similar amounts of titanium in the converter charge and of its oxide in the converter slag without any detrimental effect on blowing performance.

Variations in other elements present in blowing iron, such as phosphorus, sulphur and the residual alloys present in comparatively small amounts, do not have any direct effect on blowing performance. Low phosphorus values result from the use of high percentages of magnetite sinter in the furnace burden, and high sulphur iron is often associated with poor iron quality, thus an indirect relation to blowing performance may be evident.

The effect of iron oxide on converter slag consistency and slopping can be shown by increasing the iron oxide in the converter slag by additions of ore or scale or by overblowing. In both cases, slag fluidity is increased. Investigations to determine the reason for poor blowing performance with magnetite sinter in the blast furnace burden, caused principally by endpoint slopping, gave additional evidence of this effect. The iron oxide content of the slag from a normal Bessemer skelp blow does not exceed a total of 20 pct ferrous and ferric oxide. The relation of higher values to slopping is illustrated in table III. In (A) a value over twice as high as normal occurred in the middle of a Bessemer blow, which slopped so violently it could not be completed. In (B), total iron oxide contents of 29 to 30 pct resulted in heavy endpoint slopping on consecutive blows produced from a mixture of two Bessemer casts. In (C) the blowing performance of consecutive casts of Bessemer iron is shown, endpoint slopping on the first blow, caused by a high iron oxide content; and satisfactory performance on the following two casts. From these and other examples, the use of magnetite sinter in the blast furnace burden under certain conditions results in high iron oxide converter slags and poor blowing performance on otherwise normal blows. This condition is not consistent, occurring on individual blows only or on individual casts of iron, as in (C). The origin of abnormally high iron oxide contents has not been definitely established, but has been positively associated with the use of

TABLE III

The Effect of Iron Oxide on Slag Consistency

	Benson in Burden, Pct	Iron Analysis Si	Mn	Blowing Condition	Converter Slag Analyses SiO_2	FeO	Fe_2O_3	MnO	CaO	MgO	Al_2O_3	TiO_2
A.—MIDDLE OF BLOW SLOPPING:												
1.	26.5	1.25	0.39	Violent Slopping	40.30	47.60	0.59		0.90			
	29.0	1.22	0.40									
B.—ENDPOINT SLOPPING, CONSECUTIVE BLOWS:												
1.	35.0	1.13	0.47	Heavy Slopping, E.P.	53.16	29.38	0.57	9.62	0.60	0.45	1.80	3.64
2.				Heavy Slopping, E.P.	56.78	26.06	2.45	7.94	0.56	0.38	1.85	3.46
3.	28.5	1.10	0.38	Heavy Slopping, E.P.	57.90	21.67	7.14	6.14	0.35	0.30	2.49	3.54
C.—ENDPOINT SLOPPING, CONSECUTIVE CASTS:												
1.	21.2	1.13	0.43	Heavy Slopping, E.P.	55.40	22.48	7.26	8.18	0.50	0.30	1.94	4.18
2.	21.2	1.15	0.39	OK	61.00	16.64	6.14	8.84	0.35	0.39	1.84	4.00
3.	21.2	1.02	0.39	OK	63.40	14.32	4.29	9.49	0.47	0.40	2.14	4.30

magnetite sinter and with iron cast at lower than normal temperatures.

If blowing iron of satisfactory physical quality is produced within chemical specifications, good blowing performance should ensue. One other factor frequently creates blowing difficulty, even under optimum conditions, namely, blast furnace slag contamination by a carryover of slag in the iron ladle. This slag contains approximately 36.0 pct silica, 45 pct lime, 10 pct alumina, and 7 pct magnesia, and is present in varying amounts in iron as delivered to the converting mill. The extent to which iron is contaminated is indicated by the lime and magnesia contained in the converter slag, since these two compounds normally are present in small amounts. Of all components of the Bessemer slag, the most detrimental to slag consistency are lime and magnesia. This effect is shown in table IV. Examples of normal blowing duplex blows are shown in (A) and (B). Analysis of the non-metallic material from the surface of iron in the ladle indicated the presence of blast furnace slag, but not in a quantity sufficient to create a thin converter slag and cause slopping. In (C), 30.5 pct lime in the iron ladle slag indicated a greater quantity of blast furnace slag, and a content of 10.6 pct lime and 1.37 pct magnesia in a slag from the middle of the blow caused slopping. In this case, slopping continued through the end of the blow when an abnormal iron oxide content was created, probably because fluidity increased the reactivity of the slag. Clean blowing iron, as in the normal blow, can be spoiled by the presence of slag in a converter charge from the same cast but a different ladle, as shown in (D).

Slopping has specific detrimental effects. Not least important are the safety hazards caused by the ejection of molten slag and metal into the operating area. Production delays, added expense, and the inconvenience of cleaning up so that moving equipment can be operated are serious, particularly in times of high production. Converting mill yield and metal production losses are a direct result of slopping. The occurrence of endpoint slopping on Bessemer blows, where clean blowing is essential to accurate endpoint and afterblow determination, results in missed analyses and diversion from intended grades. The fluid slags associated with slopping in themselves are disadvantageous. The normal solid slag is retained in the converter when the blown metal is removed, and can be disposed of conveniently by emptying into a separate receptacle. Fluid slags are carried over with the metal, causing uncleanliness unless non-metallics are fluxed out. In duplex heats, quantities of this siliceous slag may be carried over into the open hearth furnace. The effect of adding unanticipated quantities of acid slag to the duplex charge is shown in table V. Compared to a normal duplex heat slag in example (A), sloppy duplex metal in (B) and (C) resulted in a high silica, low iron oxide and low V ratio duplex slag. Complete phosphorus elimination was impossible in (B) and a phosphorus reversion occurred in (C). Entry of fluid slag into the Bessemer teeming ladle decreases the efficiency of alloying additions to the ladle. An example was the necessity to increase the standard ferro-manganese addition in a 72-ton Bessemer skelp blow from 385 lb to 440 lb because of the fluid slags associated with the use of increased amounts of magnetite sinter. Metal loss on Bessemer capped and keyed blows causes short ingots, which cannot be capped and are not applied on intended orders because of inferior quality.

Slopping resulting from mechanical reasons usually can be eliminated, although not always conveniently. Reduction of the converter charge to increase the ratio between vessel volume and charge weight will permit the blowing of questionable casts, at the expense of production. Converters with excessive buildup over the lining can be replaced more frequently, improving overall blowing performance, at an increased operating cost if adequate facilities for relining converters is available. Constant attention to operating

TABLE IV

The Effect of Lime and Magnesia on Slag Consistency

Analyses	SiO_2	FeO	Fe_2O_3	MnO	CaO	MgO	Al_2O_3	TiO_2
A. NORMAL BLOWING DUPLEX: Iron Analysis: Si-1.45, Mn-0.50, $\frac{Si}{Mn}$ Ratio-2.9:1, Submarine Ladle								
Sample Identity:								
Slag From Iron Ladle	52.24	5.29	14.00	5.47	12.30	1.59	7.61	1.43
Slag From Duplex Heat	64.50	0.26	13.85	11.89	1.00	0.46	6.80	1.35
B. NORMAL BLOWING DUPLEX: Iron Analysis: Ti-1.36, Mn-0.47, $\frac{Si}{Mn}$ Ratio-2.9:1, Submarine Ladle								
Sample Identity:								
Slag From Iron Ladle	40.70	6.45	30.80	1.17	9.53	1.22	9.46	0.79
Slag From Duplex Heat	66.14	4.64	11.43	10.08	2.60	0.29	3.59	1.51
C. SLOPPY BLOWING DUPLEX: Iron Analysis: Si-1.30, Mn-0.55, $\frac{Si}{Mn}$ Ratio-2.4:1								
Sample Identity:								
Slag From Iron Ladle	48.00	2.19	1.71	1.40	30.50	3.48	12.14	0.63
Slag From Middle of Blow	60.12	2.58	9.57	8.74	10.60	1.37	5.46	1.43
Slag From End of Blow	46.76	6.32	23.28	7.16	9.80	1.20	4.46	0.93
D. SLOPPY BLOWING AND NORMAL BLOWING DUPLEX: Single Cast, Different Submarine Ladles. Iron Analysis: Si-1.37, Mn-0.60, $\frac{Si}{Mn}$ Ratio-2.3:1								
Sample Identity:								
Slag From Duplex Heat First Sub. Ladle—Normal	63.68	2.32	16.85	10.61	0.40	0.30	3.80	1.43
Slag From Duplex Heat Second Sub. Ladle—Sloppy	64.50	14.71	1.53	10.33	2.30	0.83	3.46	1.98

equipment such as blast turbines, regulators, and converter rotating mechanisms also is necessary.

Corrective measures designed to improve blowing performance by producing a converter slag of desirable character are not so easy to formulate because of the complexity of the Bessemer slag. Analyses of converter slags and corresponding fusion points are listed in table VI. Correlation of fusion point determination of slags with blowing performance has shown that this value should exceed 2900°F. The effect of various components is evident in several instances, although in all cases the fusion value is a function of several major constituents. Samples (a) and (e) illustrate the effect of high lime and magnesia values. Samples (b) and (d) are ladle slags containing excess quantities of manganous oxide, some of which originated in the ladle addition of ferro-manganese. Excess iron oxide is present in (c), (d), (f) and (g). Decreased silica resulted from the increase of these values, further increasing their harmful effect. In all but one sample, (e), silica contents over 60.00 pct were necessary to attain satisfactory fusion values. Variations of alumina and titania, in the range listed, do not seem related to the consistency of the converter slag.

Blast furnace slag can be excluded from the converter charge by complete skimming of metal in the blast furnace runner, casting into clean iron ladles, and, if necessary, skimming iron ladles at some stage of iron handling. Manganese content of blowing iron determines the amount of manganous oxide present, thus control of manganese is necessary, with some latitude permitted by accompanying variations in silicon. The presence of excess amounts of iron oxide is difficult to overcome. An attempt to correct endpoint slopping by the accumulation of slag from successive blows is shown in table VII. Often a solid residue remained after the fluid portion of a slag had been ejected in endpoint slopping. Comparison of the analyses of fluid and solid samples from a blow (A) indicates the reason for the creation of a satisfactory slag by this practice. This practice, however successful in eliminating slopping, is impractical because of difficulties related to the handling of heavy slag volumes, and the necessity for maintaining uniform blowing conditions from blow to blow. Additions of ferrosilicon to increase the silicon content of iron that could not be blown without slopping have been successful, but additions are limited because of the effect of increased silicon content on blowing temperatures. Increasing the silica content of the slag by additions of silica sand have proved effective in correcting slopping, as shown in table VII, (B), but also proved to be impractical because of difficulties in handling and adding the stand and low efficiency of the addition. During periods of blowing difficulties, the presence of blast furnace slag in the converter charge and high iron oxide converter slags appeared to be related to comparatively low casting temperatures, irrespective of iron analysis; and maintaining high casting temperatures appeared to be effective in reducing the amount of these materials present.

The problem of maintaining satisfactory converter lining life has little metallurgical significance except in an indirect fashion, since many of the factors involved also may affect steel quality adversely. Of the two directions in which the face of a lining proceeds through use in blowing metal, progressive erosion of the lining face is probably of least concern. Reasons for erosion include the various factors that tend to produce a thin converter slag, such as low silica, high manganous oxide, iron oxide, or lime and magnesia, and excessive blowing temperatures. Decreased lining life caused by erosion increases operating costs and may result in production delays. Corrective measures include adjustment of iron analysis and maintaining a high degree of iron cleanliness.

The formation of accretions on the converter lining may be occurring while erosion is taking place in some other section of the lining. So-called buildup may be distinguished as two different and distinct types, depending on the manner in which it was formed. The most common occurs in the belly of the converter and is commonly termed **kidneys** because of its form. This type of buildup is created when the converter is in a horizontal position after completion of blowing, simply by contact between the converter slag and lining with some residue remaining after each blow, if the factors that cause erosion are not active enough to remove it. Any factor that tends to increase the area or interval of contact between slag and lining aggravates kidney buildup. Prolonged delays in emptying heats because of handling difficulties and high silicon iron, which creates heavy slag volumes, are most frequently associated with the difficulty. The relation of high silicon and heavy volumes of high silica slags is indicated in the analysis of a typical kidney in a converter that blows duplex metal exclusively. Silica is abnormally high (70.80 pct) and three of the components that cause fluidity are low MnO, CaO and MgO.

The other type of buildup occurs in areas of the lining other than the belly of the converter, or areas not in contact with the bath when the converter lays horizontally, namely, the sides, back and, in aggravated cases, up to the mouth or nose. The position of this back buildup indicates that it is created during blowing or when the converter is vertical. At Aliquippa Works its existence has been uncommon, except when the blast furnace burden contained magnetite sinter. Separate analyses of three distinct strata, which often seemed to be typical of this formation were made. In each instance, from the converter lining to the blowing surface, analysis is typical of the pasty fluid slags associated with magnetite sinter and endpoint slopping, containing a low percentage of silica and higher than normal percentages of iron oxide. Apparently such slags tend to adhere to the lining during the blow, with the lower melting point constituents rejected from the surface as blowing continues.

The effect of lining buildup on blowing performance in decreasing the volume of the converter has been mentioned. Operation of the converter is often hampered by excessive buildup through the added weight alone, sometimes concentrated in one portion of the unit. The effect on costs and production rates is obvious.

The occurrence of buildup of any type is related closely to slag consistency and slag volume. Iron

TABLE V

Effect of Slopping on Duplex Open Hearth Practice

	Iron Analyses				Tapping Slag Analyses						Steel Analyses	
	Ti, Pct	Mn, Pct	Si: Mn Ratio	Blowing Condition, Duplex Metal	CaO	TiO_2	P_2O_5	V-Ratio	Actual FeO	Fe_2O_3	Prelim. P	Ladle P
A.	Normal			Normal	45.98	10.70	4.40	3.8	19.48	7.14		
B.	0.95	0.55	1.7:1	Sloppy	43.90	25.10	3.94	1.5	6.30	2.40	0.020	0.030
C.	0.90	0.50	1.8:1	Sloppy	45.35	23.70	3.94	1.7	10.20	3.10	0.008	0.035

TABLE VI

Converter Slags, Chemical Analysis and Fusion Points

Analyses	SiO_2	FeO	Fe_2O_3	MnO	CaO	MgO	Al_2O_3	TiO_2	Fusion Point, °F
a	56.30	14.32	1.14	8.00	6.95	1.56	4.87	2.20	2325
b	45.76	15.61	5.86	24.93	0.31	0.32	3.67	2.90	2800
c	51.96	16.71	14.86	6.51	0.38	0.30	5.51	3.14	2650
d	51.60	17.03	7.40	17.24	0.48	0.23	2.79	3.20	2730
e	63.90	15.09	3.00	6.62	1.80	0.87	2.52	4.40	2860
f	56.78	26.06	2.45	7.94	0.56	0.38	1.85	3.46	2870
g	60.40	19.35	5.00	8.74	0.26	0.31	1.66	3.44	2900
h	61.14	18.19	3.14	9.76	0.68	0.40	1.61	3.80	2900
i	62.00	12.50	4.80	11.68	0.70	0.19	3.80	4.20	2910+
j	63.64	11.50	8.00	8.47	0.54	0.25	3.65	3.96	2920+
k	65.64	8.53	6.64	11.04	0.67	0.31	3.43	3.96	2920+

analysis specifications, although controlled by other considerations, are established with this feature in mind, since slag volume and its character can be controlled to some extent by variations in silicon and manganese. Kidney buildup can be minimized by expediting the handling of blows out of the converter. Buildup can be reduced by the addition of agents such as fluorspar, lime or dolomite, which create fluid or erosive slags. However, this method usually is impractical since these conditions also result in slopping.

These particular converting mill problems have existed before to some degree, and they cannot be completely eliminated. The seriousness of these problems has been intensified indirectly as a result of major changes in raw materials. Some foretaste of a decline in coal and coke quality has been experienced by the mechanization of coal mines, a necessary measure, resulting in higher ash and sulphur content at present. This should be overcome by the installation of better coal cleaning facilities. Several converter plants have had experience in the use of magnetite sinters in the blast furnace burden, perhaps not to the extent that it will be necessary to use this material in the future. The effect of higher manganese iron on blowing performance is recognized and some of the precautions necessary to counteract this change are known.

It is evident that successful Bessemer converter operation in the future will depend on the concentrated and cooperative efforts of all groups related to this activity. The direction of these efforts can be suggested. An undesirable trial of a raw material might be overcome by improving it in other respects or by improving the quality of the materials that accompany it in the blast furnace burden. Blast furnace smelting, casting, and iron handling practices might be altered to overcome some unavoidable deficiency. Intermediate treatment of cast iron between casting and blowing could be accomplished. Changes in converter design and blowing practice are already an active subject.

TABLE VII

The Effect of Increased Silica in Slag

	Benson In Burden, Pct	Iron Analysis, Pct Si	Mn	Blowing Condition	Sample Identity	SiO_2	FeO	Fe_2O_3	MnO	CaO	MgO	Al_2O_3	TiO_2
A. Accumulation of Converter Slag:													
1.	49.5	1.60	0.30	Slop E.P.	Fluid	53.78	21.85	9.55	8.28	0.20	0.32	3.51	1.75
					Solid	62.96	19.30	2.32	8.28	0.20	0.46	3.39	2.85
2.	33.3	1.38	0.40	OK	Accumulated Slag	69.20	11.09	3.10	7.15	0.56	0.63	2.56	5.22
	33.7	1.52	0.40										
B. Additions of Silica Sand:													
1.	29.4	0.95	0.38	Hvy. Slop E.P.	No Sand Added	58.59	21.28	4.29	7.54	0.60	0.41	2.03	4.70
2.				Lt. Slop, E.P.	800 lb. Sand Added	64.90	15.35	3.37	7.63	0.90	0.52	2.00	4.70
3.	29.2	1.20	0.39	Hvy. Slop E.P.	No Sand Added	54.66	25.41	4.57	7.44	0.55	0.50	2.03	4.15

Faster

Open Hearth Charging

by Vernon W. Jones

AN improved open hearth scrap handling and charging technique is being built into the new No. 2 open hearth shop of the Armco Steel Corp., Middletown, Ohio, that will aid substantially in increasing the efficiency of open-hearth furnaces. By avoiding charging delays, a marked reduction in bank delays will result, and better fuel efficiency and furnace life is anticipated. The novel part of the entire installation hinges on the method of delivery of loaded scrap buggies into charging position and the removal of empty buggies from the charging floor.

The thought and consideration given furnace charging is natural because furnacemen realize that fast charging has a greater effect on furnace production than any other factor. Taylor and Wood, in a McKune Award paper presented to the Open Hearth Committee, Iron and Steel Div., AIME at the Chicago meeting, in 1946, statistically corroborated the furnacemen's often repeated remark, "You can't melt scrap in the stockyard."

In the early days of the open hearth technology in this country production was limited by many factors, the major being size of the furnaces. Furnace size was limited by the load carrying capacity of equipment of that period. As this problem was gradually overcome and furnaces were made larger than the early 7 to 10 ton units, the time consumed in charging presented a major problem. Samuel T. Wellman, Cleveland, furnished the impetus that was a major contribution to the growth of the openhearth process when he invented and installed the first charging machine at the Otis Steel Co., Cleveland, in 1888. (Patent No. 421797. Applied for Mar. 23, 1889. Granted Feb. 18, 1890). That charging machine was the first important step in the direction of faster charging practice.

The modern charging machine is the result of improvement and enlargement of the original design, but the essential features are still the same. Improvements in charging practice since that time have been along the lines of larger furnace doors, which permitted the use of larger charging pans; improved preparation of scrap to obtain more weight per cubic foot; improved facilities for getting buggies to and from the furnaces; and the use of oxygen for faster melting.

Comparison of charging practices as carried on by different plants involves a multiplicity of factors that make it difficult to decide which factors are most important. Assuming that charging facilities, such as charging machines, hot metal cranes, stockyard cranes, locomotives and operating space, are adequate, the problem resolves itself into a consideration of the conditions peculiar to each individual plant. The charging difficulties of a plant charging 70 pct hot metal are relatively small as compared to a plant using 30 pct hot metal. The plant with a high hot metal charge can use larger charging pans as there is no difficulty in revolving the pan because the furnace is not filled halfway to the roof with scrap. The plant using oxygen in the fuel can use a high percentage of light scrap and benefit by doing so.

Common to all plants is the importance of increasing to the limit the weight per cubic foot of scrap in each charging box. This factor is comparable between plants only if the weight per cubic foot of dealers heavy melting scrap is used for comparison purposes. A weight of 80 lb per cu. ft. of heavy melting scrap is indicative of good loading and leveling, and labor costs tend to get excessive when greater weight is sought.

Over the years there have been several unusual methods proposed for faster charging of open-hearth furnaces. For the most part those methods have involved mass charging, wherein it is intended to charge furnace of 150 tons or more in 10 to 20 min. The element of chance involved

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in these proposals so far have prevented any company from risking the large amount of capital involved in these radically different methods. The well known fact that under some conditions it has been possible to charge a furnace too rapidly has also tended to hold back the installation of these mass charging methods.

It may sound contradictory that a furnace can be charged too rapidly, but furnace operators have had this experience at rare intervals. This usually happens when trying to get a furnace charged ahead of tapping an adjacent furnace to avoid bunching of heats and simultaneous charging of adjacent furnaces. All too often, the rapidly charged furnace falls back into the bunch on the next round. The placing of the large mass of cold material in the furnace in a short period chills the system so drastically that considerable time is lost in bringing the system back up to operating temperatures. The success of top charging in electric furnace practice has led to numerous similar proposals for charging open hearth furnaces, but consideration must be given to the greatly different methods of melting down the cold charge in the two processes.

The No. 2 Open Hearth shop at Middletown, now under construction, is designed so that there can be no interference caused by two adjacent furnaces charging at the same time. Two conventional charging machines, buggies and pans will be used. The success of the installation is in the method of getting loaded buggies into charging positions and removing of empty buggies from the charging floor.

The delivery of buggies will be accomplished by using scrap hoists to lift the loaded buggies one at a time from the cellar, which is at yard level, to the charging floor, 21 ft above. The empty buggies will be returned to the cellar by another hoist at the other end of the furnace being charged. Fig. 1 shows the yard and charging floor levels and the buggy handling technique.

This method provides a storage track for loaded scrap buggies for each furnace. The scrap can be loaded and in position for each individual heat at a safe time before charging starts, as shown in fig. 2. Each track holds 15 charging buggies, providing adequate storage facilities.

Test runs of present furnaces in which the operation of the new No. 2 Open Hearth plant was simulated have been sufficiently conclusive to justify going ahead with the proposed charging method.

The system is designed to permit the charging of one buggy of scrap every 2 min. This will permit charging a 225 ton heat of 70 pct scrap in $1\frac{1}{2}$ hrs if the furnace will take the scrap that fast. The charging floor arrangement is shown in fig. 3. Records of actual operations show that $2\frac{1}{2}$ hrs charging time of scrap with $4\frac{1}{4}$ hrs hot metal ($1\frac{3}{4}$ hrs after finish scrap) will produce 18 tons per hr with 40 pct hot metal in a furnace with a 568 sq ft hearth area. (180 tons per heat). This, of course, was possible only with ideal charging conditions.

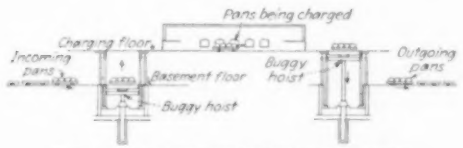

Fig. 1—The yard and charging floor levels and the buggy handling technique are shown here. Buggies are raised to charging floor level by hydraulic hoists.

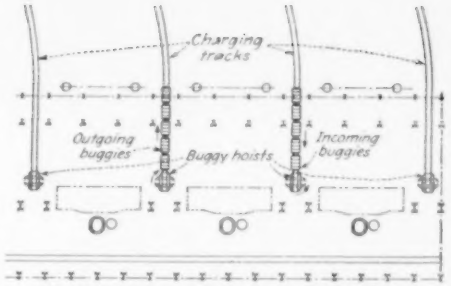

Fig. 2—Each track on a charging floor holds 15 buggies, providing adequate storage facilities. The system permits charging one buggy of scrap every 2 min.

An advantage gained from this layout will be the fact that the stockyard can be operated at an even rate. There will be no periods in the stockyard when pans must be loaded as fast as possible, with the resultant bad loading and trimming; following which is the usual slack period when the stockers have nothing to do. By having enough charging buggies in service, a steady flow will come out of the yard and be stored, in station order, on the ladder tracks to the scrap hoists.

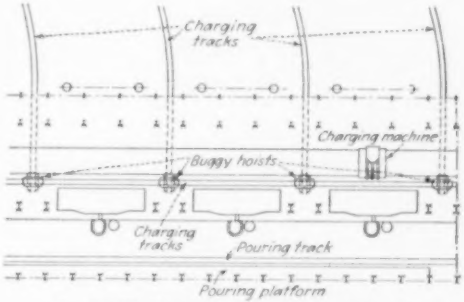

Fig. 3—Moving toward the furnaces on the charging tracks the buggies are transferred to the charging track and picked up by the charging machine. The system permits stockyards to be operated at an even rate and reduces bad loading caused by big demand.

The General Engineering Dept., of Armco Steel Corp., is the group responsible for this method of delivery of scrap to the furnaces. It is believed that this layout, which solves several operating problems of an open hearth dept., will be a major foreword step in increasing the efficiency of open hearth furnaces. By avoiding the charging delays a marked reduction in bank delays, will result in better fuel efficiency and furnace life. No difficulty is anticipated in producing the 400,000 tons per year for which this shop is designed.

Modern Blast Furnace Gas Cleaning Practices

by G. P. Burks

BLAST furnace gas cleaning has progressed through many stages of development since top gases were first collected and utilized for heat value. The methods for separating dust from the gas stream are almost innumerable. With modern hot blast stoves, coke ovens, boilers, gas engines and metallurgical processes requiring large volumes of fine cleaned blast furnace gas, this problem assumes great importance.

At Gary Works of Carnegie-Illinois Steel Corp., the trend has been toward larger and more efficient primary dust catchers, followed by static primary washers and, for fine cleaning, the electrical precipitator. At present the gas system is being revamped and supplemented by new equipment to provide additional fine cleaned gas to fire new, high efficiency boilers, to underfire coke ovens and to replace outmoded equipment.

While not affecting the blast furnaces proper, basic changes are underway in the gas utilization system. To follow these trends, the Gary system, as it was prior to the changes, and the current program will be described.

1—The Blast Furnace Plant consists of 12 furnaces with facilities arranged so that in final analysis there are six groups, of two furnaces each, in line.

2—All gas is primary washed at the blast furnaces, with each pair of furnaces having its own cleaning system, except Nos. 1, 2, 3, and 4 furnaces which have individual facilities.

3—Where fine cleaned gas is required for hot blast stoves, each pair of furnaces has its own fine cleaners.

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4—Excess primary-washed gas is discharged by each pair, as the case may be, into a collector main that runs parallel to the furnace line.

5—Also arranged in three parallel lines to the furnace line are three Theissen washer houses (eight units each); three gas-engine blowing rooms; and four gas-engine power stations.

6—The Theissen washers take primary-cleaned gas from the collector main and furnish fine cleaned gas for gas blowing engines, and also gas-engine driven generators.

7—The remaining primary gas is used to fire boilers, with the excess being burned at three automatic bleeders spaced equally along the line.

The modernization program for gas utilization includes two plate-type electrical precipitator units, installed to replace one of Theissen washer houses; and three large, high-efficiency boilers, installed to burn fine cleaned gas. Nearing completion are four additional tube-type electrical precipitators, two which will clean primary gas and two which will take gas from the fine cleaned gas main and further clean it for under-firing coke ovens. In addition, work is underway to construct two more plate type electrical precipitators to clean primary gas. When completed only one Theissen washer house will remain. Ultimately, however, replacement of the last Theissen washer house can be expected.

Table I summarizes blast furnace gas cleaning units and their performance, and table II gives similar data for the Power and Fuel Div., gas utilization system. Rather than duplicate descriptions and illustrations, only one of each type unit will be considered.

Fig. 1 illustrates a primary tower washer recently installed on No. 2 blast furnace. This design typifies the trend at Gary Works toward a simplified static tower washer for primary clean-

ing gas. Each bank of tile consists of a 12 in. thick random layer of 2 x 2 x ¼ in. tubular tile. In tracing the gas flow up from the inlet to the outlet of the washer the following stages are passed:

1—Two tile banks, 2—Spray chamber fed by service water, 3—Three tile banks, 4—Spray chamber fed by high pressure water, and 5—Strip baffle type of moisture eliminator.

Three primary washers on Nos. 5 and 6 furnaces have identical arrangements and differ only in size. The tile banks, which are of columnar design, have their largest openings in the lowest bank. These openings are made increasingly smaller through the two successive pairs of tile banks. From inlet to outlet these washers have the following stages: 1—One 12 in. tile bank (rings), 2—Spray chamber, 3—Two 18 in. tile banks (partition rings), 4—Wetting chamber, and 5—Two in. tile banks (drip tile).

Glass wool packing is shown on top of the two upper tile banks. This material is no longer in these washers. Although this packing apparently was doing a good job, it wasn't strong enough and, in failing, packed up in some areas and blew out in others. This caused a channeling of the gas which necessitated the removal of the glass wool.

Fig. 2 shows the electrical precipitator used to fine clean gas for the hot blast stoves on Nos. 5 and 6 furnaces. This is a conventional tube-type unit with a rated capacity of 85,000 cfm of primary cleaned gas, and is divided into two compartments, each of which contains 130 12 in. x 15 ft. collector tubes.

In typical arrangements of electrical precipitators, discharge electrodes in the form of wire or rods are connected to high voltage uni-directional current, and the collector electrodes, either tubes or plates (depending on precipitator type), are grounded. The field set up between the two permits ionization of the dust particles drawn to the collector electrodes. The film of water flowing over the electrodes carries the particles away. Entrained moisture is eliminated in the same manner. The degree of elimination is illustrated by the dehydrating action on wooden plugs driven into clean out and purge holes on the clean gas main. After the installation of a precipitator, the plugs shrank, blew out and were replaced by pipe nipples and caps.

Fig. 3 shows the gas washing systems for Nos. 7 and 8 furnaces, which includes three tower washers for primary cleaning. These units utilize individually driven rotors, consisting of conical vanes arranged in concentric clusters. Upon being rotated, these rotors lift water from a pan and discharge it in the form of rain and mist from the periphery of the vanes. The stages in these washers from inlet to outlet are as follows: 1—Two stages of rotors, 2—One 12 in. thick bank of 2 x 2 x ¼ in. tubular tile, 3—Spray chamber, 4—One 12 in. thick bank of 2 x 2 x ¼ in. tubular

Fig. 1—Details of the primary washer tower recently installed on the No. 2 blast furnace. This tower typifies the trend at Gary to a simplified static tower washing for primary gas cleaning.

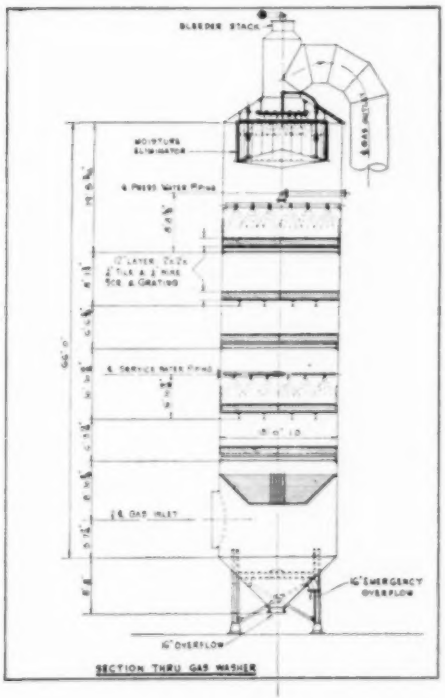

Fig. 2—Details of the electrical precipitator used to fine-clean gas for the hot stove blasts is a conventional tube-type unit with a rated capacity of 85,000 cfm.

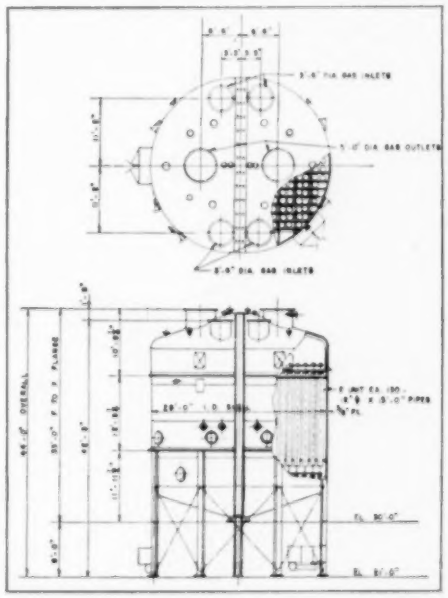

TABLE I

Blast Furnace Gas Cleaners

Furnace No.

PRIMARY TOWER WASHERS	1	2	3	4	5 and 6	7 and 8	9 and 10	11 and 12
Number and type	1—Static	1—Static	1—Rotor	New furnace under construction. Will include new primary washers and a tube type electrical precipitator to clean stove gas.	3—Static	3—Rotor	2—Rotor	1—Static, 2—Rotor
Number and size	1—18 x 66 ft	1—18 x 66 ft	1—18 x 57½ ft		2—18 x 62½ ft 1—20 x 62½ ft	3—18 x 62 ft	2—20 x 58 ft	1—18 x 77 ft-10⅜ in 2—18 x 69 ft-8⅜ in
Capacity per unit, Cfm	80,000	80,000	80,000		80,000 each	80,000 each	80,000 each	80,000 each
Gas cleaned per unit, Cfm	74,000	74,000	74,000		64,500	71,000	96,000	65,500
Moisture Eliminator	Built in	Built in	Separate		None	Built in	Built in	Built in
No. of tile banks	5	5	3		5	2	2	7—Static, 3—Rotor
No. of rotors	0	0	1		0	2	2	0—Static, 2—Rotor
Pressure drop (avg.), In.	15	15	15		15	15	15	15
Avg. gas cleanliness Before: gr per cu ft	4 to 5	4 to 5	4 to 5		4 to 5	4 to 5	4 to 5	4 to 5
After: gr per cu ft	0.1229	0.0863	0.1184		0.1185	0.1031	0.1556	0.1132
*Water consump (calc.) Gal per day	3,200,000	3,200,000	3,200,000		2,800,000	3,130,000	3,200,000	2,930,000
*Power consump (est.) Kw-hr per MM cu ft of gas	0	0	25 gas		0	25	25	Rotor—25
SECONDARY CLEANERS	None	None	None					
Type					1—Precipator (2 compartments)	2—Disintegrators	2—Disintegrators	1—Precipitator (2 Compartments)
For precipitator type of collector					Tube			Plate
Capacity					85,000	30,000	30,000	80,000
Gas cleaned per unit					50,000			50,000
Moisture Eliminator					None	2—Separate	2—Separate	None
Pressure drop (approx.), In. W.G.					0.25	5	5	0.25
Avg. Gas cleanliness-inlet gr per cu ft					†0.200	0.1031	0.1556	‡0.200
Avg. Gas cleanliness-outlet gr per cu ft					†0.010	0.0327	0.0700	‡0.010
Water Consumption—Normal (est.) Gal per day					144,000 per comp	200,000	200,000	144,000 per comp
Water Consumption—Flush (est.) Gal per Min per Comp.					750			750
Power Consumption (est.), Kw-hr per MM cu ft					3	116	116	3

* Each washer.

† Manufactures rating at 85,000 cfm gas.

‡ Manufactures rating at 80,000 cfm gas.

While neither precipitator unit is required to clean its rated volume of gas, both exceed considerably their fine cleaned gas rating.

tile, and 5—Strip-baffle type of moisture eliminator.

Stove gas for these two furnaces is further cleaned by two rotary disintegrators wherein water is ejected radially by centrifugal force against rotating bars set in three concentric rows at the periphery of the rotor and parallel to its axis; and, also, against stationary bars independently mounted. As the gas stream must flow through the spaces between the bars, the dust particles are wet in the misty and turbulent passage, thus dropping out of the gas. Each disintegrator has its own moisture eliminator consisting of a large chamber with three 8 in. banks of $1\frac{1}{4}$ x $1\frac{1}{4}$ in. pipe nipples. Spray systems are located above each bank. The pipe nipples proved unsatisfactory because of the tendency to corrode and plug up. These are being replaced by the standard 2 x 2 x $\frac{1}{4}$ in. tubular tile.

The Theissen washers mentioned in connection with the Power and Fuel Div. gas cleaning system consist of a rotating drum with helical vanes attached to the outer surface of the drum. A fan wheel is mounted on the end of the shaft at the outlet to boost the gas pressure. Normally, the inside of the housing is covered with a coarse wire screen, but at Gary the screen has been replaced by bars. In operation, the gas and water are thrown against the casing, assuring intimate contact. A favorable characteristic of this type unit, which includes the disintegrator, is that within reasonable limits, there is no apparent relationship between the dust load in the inlet gas and the cleanliness of the outlet gas.

On the blast furnace side is a two compartment plate-type electrical precipitator rated at 80,000 cfm of primary washed gas. This unit is located between Nos. 11 and 12 furnaces and cleans gas for the eight stoves serving these furnaces. Fig. 4 shows the general layout of the precipitator unit, and table III contains general information relative to the unit.

Each precipitator compartment has its own power unit enclosed in a compact cabinet located in an adjacent building and includes transformers, mechanical rectifier, control panel and ventilation facilities. The transformer for each power unit is rated at 25 kva with a 440 v. primary and 35,000 to 65,000 v. secondary.

Table II
Power and Fuel Div. Gas Cleaning Equipment in Operation

Theissen Disintegrators

1—No. of Theissen Disintegrator Houses—two
2—No. of units per house—eight
3—Rated capacity of each unit—15,000 cfm
4—Gas cleaned per unit—15,000 cfm
5—Pressure boost—3 in. W.G.
6—Average gas cleanliness, Inlet—0.110 gr per cu. ft.
7—Average gas cleanliness, Outlet—0.011 gr per cu. ft.
8—Water consumption—122,400 Gal per Day
9—Power consumption—150 kw-hr per MM cu. ft.

*** Plate Type Electrical Precipitators**

1—No. of units (two copartents per unit)—two
2—Rated capacity per unit—94,000 cfm
(†) Gas cleanliness, Inlet—0.15 gr per cu. ft.
(†) Gas cleanliness rating, outlet—0.015 gr per cu. ft.
4—Pressure drop—0.25 in. W.G.
5—Water consumption, each compartment:
Normal—129,600 Gal per Day
Flushing—275 Gal per Min.
6—Power consumption (est.)—3 kw-hr per MM cu. ft. gas

* No indicative test results available at this time.
† Manufacturers rating.

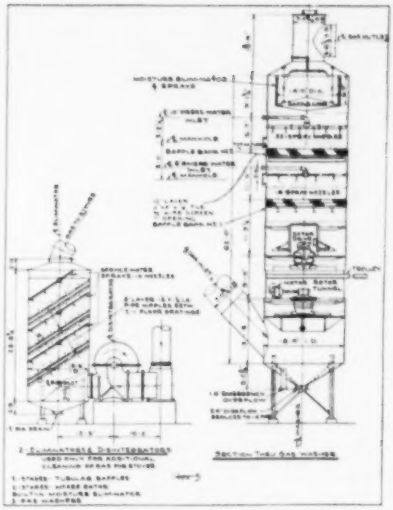

Fig. 3—The gas washing system for Nos. 7 and 8 furnaces includes this system of three tower washers for primary cleaning. These units use individually driven rotors consisting of conical vanes arranged concentrically.

An important safety feature of each unit is an interlocking key block system. Each precipitator compartment entry door has a lock and key arrangement, and each door must be closed and locked to free its respective key. Each key must be inserted simultaneously into a master key block to free the master key. The master key in turn is removed from the block and used to unlock the ground switch. Only when the ground switch is opened can the control circuit be set up and the unit placed in operation. Conversely, only when the ground switch has been closed can the master key be freed from the power unit and used to free the keys to the precipitator compartment doors.

Only one serious weakness has been noted in this unit, and that has been corrected. This concerns the discharge electrodes which are 5/32 in. square stainless steel rods supported and aligned in rows of nine by pipe frames. These vertical rods were held in alignment by placing them through holes drilled in the pipe frame. At the top, the rods were welded to the pipe frame for support, but fit loosely through the pipes aligning the middle and bottom. This loose connection resulted in localized arcing between the rods and the pipe, which gradually eroded the rods until some broke. In leaning over on the collector plates, the rods shorted out the unit. By welding the rods to the pipe frames at each point where the rods pass through the frame, this condition has been corrected.

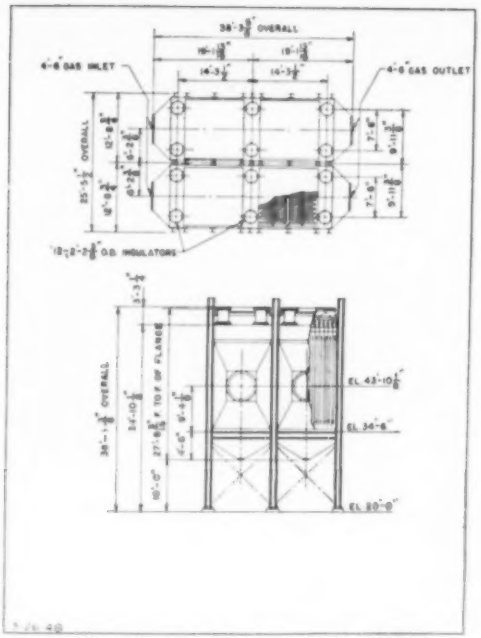

Fig. 4—The general layout of the precipitator unit is shown here and details relative to the unit are included in table III.

Table III

Plate Type Precipitator Used on Nos. 11 and 12 Furnaces

Precipitation Chambers:

Number—two
Size—11 ft 8 in. wide x 38 ft $3\frac{5}{8}$ in. long (overall) x 27 ft $8\frac{5}{16}$ in. high

Collector Electrodes (Plates):

No. per chamber—52 (104 total unit)
No. of rows per chamber—13 (plus two wall plates)
Size of plates 20 ft x 5 ft x $\frac{5}{16}$ in.

Weirs:

No. of weir boxes—one for each plate.
Size—$3\frac{1}{4}$ in. wide x $4\frac{1}{2}$ in. deep x 5 ft long.

Discharge Electrodes (Pipe frames hold vertical discharge wires):

Size of $\frac{3}{4}$ in. pipe frames—4 ft $8\frac{1}{2}$ in. x 17 ft $4\frac{1}{8}$ in.
No. of frames per chamber—56 (112 total unit)
No. of rows—14
No. of frames per row—four
No. of wires per frame—9 wires, 6 in. apart.
Size of wires—$\frac{5}{32}$ in. sq. x 17 ft $4\frac{1}{8}$ in. long
No. of wires per chamber—504 (1008 total unit)

Feed and Discharge:

Size of mains—6 ft
Goggle valves—four 54 in.
Baffles installed at entry of each compartment to create uniform gas flow through the precipitation chamber.

Gas Velocity:

Average velocity of gas through unit at 50,000 cfm —2.20 fps.

Average velocity of gas through unit at 80,000 cfm —3.50 fps.

Another difficulty experienced when the unit was first put into operation concerned the baffle plates installed in the inlet. The anchorings that held these plates in place were not strong enough and in failing, permitted some of the baffle plates to swing in the gas flow, thus making contact with the electrodes and shorting the unit. The nature of this trouble made it difficult to locate, but once located, readily corrected.

One of the first difficulties experienced was with the rectifier rotor. The mount and bearing assembly was not strong enough, resulting in the rotor getting out of synchronism, which in turn caused commutation difficulties. This trouble was noted early, and with modification of the design, the rotor has since operated trouble free.

In operation, this unit is flushed twice a day by opening the flushing valves wide and leaving the irrigation water (normal water to produce flowing film on collector plates) at the standard setting. This flushing period lasts until the flush water becomes clear.

The irrigation water is regulated by ordinary gate valves. The proper amount was determined by observing the water film on the plates while the covers were off and by noting the valve settings and discharge appearance when optimum flow was reached.

Other than flushing the unit and checking the electrical settings twice a day, the only attention necessary has been routine checking and cleaning of the electrical equipment and periodic inspection and cleaning of the stainless steel weir boxes. As in any weir system, the non-colloidal solids in the water supply will settle out and build up.

This was the initial installation of a plate-type precipitator for cleaning blast furnace gas. After the weaknesses were corrected, the performance, maintenance and operation have been entirely satisfactory. From Gary Works experience, it is believed that electrical precipitators, either the tube or plate type are satisfactory for the fine cleaning of blast furnace gas.

With a thought to the future, an improvement of the methods of dry cleaning gas possibly would be helpful because of the inherent, unquestioned advantages of eliminating usage of large volumes of water, and the consequent recovery of sludge, and in some cases, stream pollution. A step in this direction might be the development of high temperature filters to simplify the highly efficient, though expensive and delicate, European Halberg-Beth system, which at present utilizes cotton filter bags. Another development, now in the research stage, is based on ultrasonic methods of agglomerating dust particles in gas streams. These and other developments will be of great interest to blast furnace operators and engineers interested in more efficient and cheaper methods of fine cleaning blast furnace gas.

Blowing Out Carbon Hearth Furnaces

by W. R. Zehner

THERE comes a time in the operation of every blast furnace when because of lower demand for its product or the need for general repairs, the burden in the furnace must be removed. This burden, consisting of coke, ore, limestone, scale, etc., will amount to 800 to 1000 tons in a 25-ft hearth furnace. Coupled with removing the stock is the question of whether the salamander in the hearth is to be removed. Until the advent of carbon hearths, furnaces were generally operated until a general rebuild, including a new hearth, became necessary. There were numerous instances where a furnace was blown out because of business conditions, when the "down" period was so long that it did not pay to bank the furnace. Now, with carbon hearths in wide use, it becomes a matter of prime interest to all blast furnace operators to be able to do a satisfactory job of blowing out such a furnace without doing any damage to the carbon lining.

Carbon, used as a refractory in a blast furnace hearth, has been performing an excellent job. However, as with any material, whether used for refractories or other purposes, it is necessary to be aware of possible dangers of spoiling it or making it lose its effectiveness. Moisture is the one thing that will ruin carbon as a refractory. Moisture in contact with carbon will make it soften and "punky," lowering its strength, density and refractoriness. Thus wastes the money spent on the installation, as the sole purpose of carbon is to furnish a refractory that will stand high temperatures and erosion. In an operating furnace this water might come from leaky tuyeres, coolers, cinder notches, cooling plates or hearth stave coolers. In blowing out a furnace with carbon hearth and side walls, the water may come from any of these sources, or it may be water added through the tuyere openings, to chill the stock while raking it out.

During the past six months, Bethlehem Steel Co., at Bethlehem, Pa., has blown out two carbon hearth furnaces. One furnace of 27 ft 2-in. hearth diam, with 881,118 tons on the carbon hearth and 2,296,971 tons on the stack, was blown out to reline it from the top of the double-wall carbon hearth up to the top of the stack, including wearing-plate section, offtakes and downcomers. This furnace, "C," was blown down June 21, 1949.

Because this furnace had produced 881,118 tons on the carbon hearth, it was thought desirable to tap the salamander at the front of the furnace under the tap hole at the original carbon floor level to expose the side walls and hearth. The front cast-iron cooling stave section where the oxygen lance burned through had to be replaced and the hole through the carbon side wall blocks had to be repaired by replacing the carbon block affected.

The other blowing out experience was on Dec. 29, 1949, on a 19-ft diam carbon hearth furnace that also needed a new stack lining. It had produced 831,898 tons on the carbon hearth. In this case, the salamander was not drained.

In both of these blowouts the same procedure was followed, and the operation was without incident. The recording instruments used were the usual blast pressure gages, top-pressure gage with a high enough range that a heavy slip would not strain the pressure element, water meter on the water going in at the top, the regular top temperature recorder in the downcomer, plus two additional recorders in the bleeder pipes, and an air-operated feeling rod hung from a chain long enough to reach from the mantle height inside the furnace up through the feeling-rod hole and down over the side of the furnace—some 20 to 30 ft. This chain was used to minimize the chance of losing the rod because of high temperature. The additional top temperature recorders were used in the bleeder pipes, because the regular recorder would not show the correct temperature after the furnace had been sealed off at the washer towers. It is necessary, in blowing out a furnace, to install a positive plate seal in the gas lines at the washers and to take a short delay to install the special feeling rod, the spray water

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pipe and to tie down the big bell. It was decided that it was safer to do this job during the daylight hours. This also made the critical or potentially dangerous period come on the day shift when more supervision was possible.

In blowing out any furnace, a number of different methods of filling the furnace for the last 12-hr period may be used. The burden may be completely replaced with nut coke ($\frac{3}{4}$ x $1\frac{1}{2}$ in.) and the wind kept on until this nut coke gets down to the tuyere level. The furnace is kept full of nut coke all the time. This material is then flushed out as in a hydraulic mining operation. Another method is to lighten the burden to about 50 pct of the normal amount of ore and stone, and charge a heavy ore blanket on top of the stock column, then stop filling and let the burden work down without any further charging. By this method it is possible to get the stock all the way down to the mantle, and rake out remaining material by hand. This method was used.

In these blowouts, the burdens were all lightened, scrap was removed the day before, and the dust catchers were constantly kept empty. Since there was enough gas available for the stoves, they were kept on the hot side in case a bosh slip should peel off some material that would chill any of the tuyeres.

The ore blank of 100,000 lb on top of the stock was put in to tighten up the stock column and discourage "blowing through," and also to hold down the top temperature. The 20,000-lb blank on the big bell was used to seal any gas leaks between bell and hopper and to soak up as much of the heat as possible. One of the limiting factors in blowing out a furnace is top temperature. Normally the fresh stock being charged will keep the top temperature down; in blowing out there is no fresh stock being charged so that the temperature will gradually rise until there is danger of warping or buckling the bells and hopper. It is at this point that the furnace is cast and sealed off at the washer towers, water sprays are introduced at top of furnace and the last stage starts.

In a plant such as Bethlehem's, with gas-driven blowing engines and gas-driven electric generators, any hyrogen gas, resulting from dissociation of water in the furnace, will cause the engines to backfire. Only limited amounts of water must, therefore, be added before the furnace is sealed off the main gas line. From this point on until the last cast, all of the gas produced goes out at the top through the furnace bleeders. The dust catcher is kept full of steam, and steam is kept between the large and small bells.

In the event of any delay which would take the wind off the furnace, it is most important that the water sprays going in the top of the furnace are immediately shut off. This task must be assigned to some responsible person who will be on the job all the time. As the stock in the furnace moves down, steam will appear around any cooling plate openings in the stack of the furnace shell. If the stack lining is not to be torn out, the blowing out must stop before the top temperatures get too high or the passage of steam through these openings becomes too rapid. In each of the two furnaces mentioned above, a point was reached about 15 ft above the mantle with top temperatures reaching 1000 to 1100°F.

After the last cast is made, all the tuyeres are clayed back; and the openings where the stock is to be raked out—on opposite sides of the furnace—are prepared with chutes that will discharge directly into hopper cars for salvaging and recharging into another furnace. It is dangerous to open too many tuyeres to rake out the stock, as the stock column may make a slip, which is very hazardous to the men.

To prevent gas from working back through the tuyeres not used at the start of the raking out process, substitute blowpipes are made up with a $1\frac{1}{2}$-in. machined disk on each end, welded on to a 3-in. double X pipe. This makes an exact replacement for the blowpipes. Over the bustle pipe openings, where the raking is to start, the bootlegs and elbows were dropped and 1-in. plate blanks were bolted on the openings. This makes it impossible for any gas to get through the bustle pipe back to the stoves or blowing engines.

As the stock clinkers and blocks the openings it may be necessary to dynamite some of the pieces to get them through the cooler openings. The stock will gradually stop running and it becomes necessary to open adjacent cooler openings until the angle of repose is reached and no more stock can be raked out. From this point on, men must get inside the furnace and shovel the stock out through the cooler openings or through a large opening in the bosh and tuyere jacket that is removed for that purpose.

Blowing out a furnace is a procedure that can give lots of trouble. A close check must be kept on all jobs, who is responsible and when the different steps are to be carried out. A complete log sheet should also be kept for the record.

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by C. P. Johnson

WHEN the "A" furnace at the Federal Furnace Plant of Interlake Iron Corp. was rebuilt in 1948, new shell steel and top structure were provided, and a carbon hearth was constructed by laying 16 blocks of carbon, 30 in. thick x 24 in. wide on a ceramic base of 18-in. bottom blocks. There was a 2-in. space left between the blocks, into which was rammed a carbon paste prepared by the Great Lakes Carbon Co. The carbon blocks on the furnace bottom had a diameter of 19 ft 6 in. and extended to within about 4 in. of the ceramic lining on the inside of the cast-iron cooling staves. This 4-in. space was also rammed with carbon paste. When the bottom carbon blocks were in place with all seams rammed with carbon paste, the outer surfaces were ground level so vertical carbon blocks would be plumb.

The wall of the crucible was made up of 31 car-

bon blocks, 29 of which were of the identical dimension, having an inner face of 20 23/32 in. and a back dimension of 23 25/32 in. The blocks were 9 ft 3 in. high and 15 in. thick. The two blocks not conforming to these dimensions were located at the tapping hole where the two faces meeting at the tapping hole had no chamfer. The two carbon blocks at the cinder notch were formed to allow a 4½-in. circle of fire brick to surround the cinder notch cooler.

Before these blocks were set, the three special carbon blocks, which fit against the cast-iron cooling staves at the tapping hole, were put in place and fire brick was fitted around them on three sides. These special blocks were 12 5/16 in. thick and together stood 6 ft high. The two large carbon blocks completing the tapping hole section were then smeared with carbon cement and set against the three smaller blocks. The 29 remaining blocks were then wedged into place so that they formed a 17 ft 2 in.-circle. The space behind the blocks was about 4 in. and was rammed with carbon paste. There was a 10½-in. fire brick wall against the cast-iron cooling staves and this formed the back wall for the 4 in. of rammed carbon paste.

C. P. Johnson is General Superintendent of Interlake Iron Corp., Chicago. *This paper was presented at the Blast Furnace and Raw Materials Conference, April 10-12, 1950, at Cincinnati.*

Before this furnace was blown in, a protective wall of arch brick 4½ in. thick was laid against the side walls of carbon blocks, whereas the bottom carbon was protected with two courses of 3-in. fire brick.

This furnace was blown in on Dec. 14, 1948 and blown out for economic reasons on July 12, 1949.

About a month after being blown in, a hot spot developed on the south side of the furnace and water was applied to the shell. Subsequent drilling through the steel shell revealed an absence of brick lining over an extensive area. Consequently in addition to the economic reason for taking this furnace out of blast, there was the desire to see what repairs were necessary. Because the hearth and bosh were so new and the hearth was of carbon, it was decided to add water as sparingly as possible when cooling the furnace off preparatory to raking out the stock.

The furnace was blown out at 3:15 PM on July 12 and the coolers were removed for raking out the stock at 10:00 AM on July 15. In the meantime, water was sparingly admitted through the try rod holes and when any water appeared at the tuyeres, all the sprays were shut off until the water dried up. The water consistently showed up at the south side tuyeres and not on the north side of the furnace. The five coolers on the south side of the furnace were pulled and raking out commenced at these openings only.

As the raking out progressed, considerable hot coke rolled down from the north side of the furnace and appeared at the south side tuyeres. This material was cooled as it appeared with hoses from the cast house floor, rather than by turning on the try rod sprays.

On July 17, two days after starting to rake out, the stock was sufficiently removed so that the furnace was vented. At this time steam was shut off the dust catcher and the three north side tuyere coolers pulled. The coke and stock removed from these openings was still quite hot, but was all removed by July 19 to tuyere level. At only one time during this raking out period did any water show up at the tapping hole and when it appeared, all water was taken off until it had dried up.

Inspection of the lining indicated that an in-wall relining was necessary and the lining to the top row of bosh plates was subsequently removed.

After removal of the brickwork, work was resumed digging the coke out of the hearth. It was possible, by using jackhammers, to penetrate the mixture of slag and coke in the crucible to within a foot of the original hearth floor, which was 4 ft below the center line of the iron notch. There was a crust of coke, slag and limestone approximately 4 to 6 in. thick covering the bosh around the tuyere coolers and extending downward over the carbon side blocks to the floor of the hearth. The tapping hole was full of this mixture also. When the crust had been broken and pried off the side walls, it was evident that whereas the side wall carbon blocks were exactly as they were when the furnace was blown in, the tapping hole carbon side blocks were eroded away in a funnel-shaped manner extending across the entire width of 4½ ft of the two blocks and going into the two side blocks adjacent to the two blocks at the tapping hole. This erosion was not the result of any procedure followed in the blowing out of the furnace, but had taken place during normal operation.

The 12 in. of material on the furnace bottom, which could not be removed with jackhammers, was drilled through in numerous places and found to be iron. After it had been penetrated, the drill again struck soft material into which a steel point could be driven, indicating that it was the original hearth floor. Readings taken at all points drilled indicated that the carbon had not worn away in any place.

The tapping hole was repaired by replacing a portion of the carbon blocks with fire brick. The protective brick was again placed against the inside face of the side wall carbon blocks and in December, 1949, this furnace was blown in.

Performance of High Magnesia Ramming Mixes

by Rudolph Smith

PREFERRED material and practice in the installation of open hearth bottoms are controversial subjects. To add to the complexity of the subject a new factor in the form of high magnesia content ramming mix has been added within the past few years.

The open hearth shop at the Pueblo plant of Colorado Fuel & Iron Corp. consists of 16 furnaces tapping an average of 110 tons. Normal operations call for 48 pct hot metal charge in making steel to specifications ranging from 0.5 to 1.00 pct carbon with 75 pct of the total production being around 0.70 pct carbon because of the company's rail and rail-fastening business. Being located in an agricultural district, much of the scrap is country, auto and bundled grades. These conditions have a definite bearing on open hearth furnace bottom maintenance.

To maintain ingot production at the highest level during the past decade, attention was focussed on bottom delays. Replacing bottoms only as often as is economically practicable does not result in minimizing delays. Experience indicates that furnace bottoms deteriorate rapidly and most shop records will produce evidence where delays on an old bottom run lower than those on a relatively new one. In this shop of 16 furnaces, 30 new bottoms were put in during the past 20 years. Of these, four were rammed in and the remaining were sintered in magnesite. Because of this experience, there developed the question of what might be the result as regards better furnace bottoms if the magnesia content of the material was increased to the highest degree.

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This was first attempted by reducing the amount of slag used for sintering in the magnesite. Records show that 35 pct slag was used in sintering in No. 16 bottom in 1927, while the last sintered in bottom of No. 7 furnace was put in with less than 10 pct slag in 1946, the magnesia content being increased from 56 to 74 pct and the lime content reduced by half to 6 pct.

With thinking on this subject along the lines of a higher magnesia content material, the proposal to try such a mix developed by one of the refractory manufacturers was acceptable. This occurred during the early part of 1947. The material was tried on a number of minor hot repair jobs with indications that it compared satisfactorily with other materials used for similar purposes. Then followed a number of trials using the material for renewing the tap hole section during the rebuilding of the furnace. These trials also showed the material to be satisfactory. The characteristic of the material to set up hard without being subjected to a fusing temperature was of considerable importance.

The first carload of the high magnesia content ramming mix was purchased in September, 1947, and was used throughout the shop for the same purposes as the regular ramming mixes. Records of delays on tap holes, which were put in with the two kinds of ramming mixes, during a particular period were: No. 15—1:20 and No. 13—2:05 with high MgO; and No. 9—1:20 and No. 11—1:55 with the regular mix. The two mixes performed equally well.

This information, plus the satisfactory performance of the high magnesia ramming mix used to repair banks at slag line during furnace rebuilding, led to the decision to use this material in putting in a new bottom in No. 8 furnace. However, because of the caution for which many open hearth people are noted, it was decided to ram in the banks to final contour, but sinter in magnesite on the flat of the bottom. The first round of magnesite was put in on May 6, and the first heat was tapped on May 10. There wasn't very much time nor money saved on this job, but information was still being sought on the performance of the high magnesia ramming mix. During the 96-hr period of sintering in the magnesite, the furnace was maintained at the highest temperature the roof could stand. The banks being subjected to this temperature showed no failure.

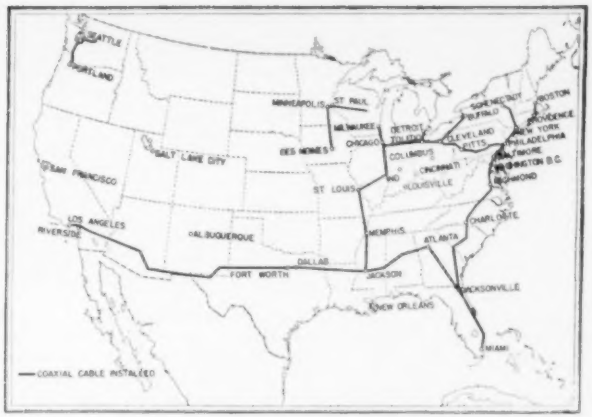
To compare the performance, records of bank delays on No. 8 and No. 7 furnaces were compiled over a period of 16 months following the installation of the new bottoms, No. 7 being all sintered in magnesite. The bank delays for No. 8 furnace with the high magnesia ramming mix averaged 4.1 min per heat, while those for No. 7 furnace, with magnesite, averaged 5.2 min per heat.

Following the steel strike, a new bottom was placed in No. 9 furnace. It was rammed in with high magnesia material and about 3 in. of magnesite was sintered in on the flat. A similar job was done on No. 16 furnace, but a regular ramming mix was used. Both furnaces set a production record in January, that for No. 16 being an all-shop individual furnace record. No. 16 operated at a rate of 12:6 tons per hr and No. 9 at a rate of 12 tons per hr, which is not bad performance for furnaces with a hearth area of about 500 sq ft. No. 16 is slightly larger, an end furnace.

Comparing the bottom, bank and tap hole delays for the first roof campaign, those for No. 9 are about 35 pct higher than for No. 16, which averaged 12.2 min per heat. Even though No. 16 is credited with being of superior design and having better crews, the performance of the high magnesia ramming mix has not equalled that of the regular mix in this particular trial.

Since Mar. 1, three furnaces have been put into operation, the bottoms of which were surfaced with different brands of ramming mixes. One is a high magnesia material. In each case, prior to shutting down for rebuilding, the furnace was washed out, removing about 6 in. off the flat and as much as possible off the banks.

The Manufacture Of Coaxial Cable



About 5000 miles of coaxial cable are in service, providing transmission between the points shown.

by W. E. Hammersla

THE manufacture of coaxial cable represents the application of mass manufacturing techniques to a product of laboratory requirements as to dimensional tolerances and raw materials. The wide band carrier system employing coaxial cable is used to transmit simultaneously either hundreds of telephone circuits or two television pictures over each pair of coaxial units. The cable must be extremely uniform in its electrical behavior, particularly in the case of television transmission.

In practice, four, six or eight coaxial units are stranded together with accompanying wire circuits into a cable which is called coaxial cable. This cable, by being capable of transmitting sufficient carrier frequency bands or channels, can accommodate 600 simultaneous conversations over each pair of coaxial units making up the cable.

The coaxial unit is a four-part assembly consisting of a center conductor of semi-hard copper wire approximately 0.1 in. diam upon which, at intervals, are spaced polyethylene disk insulators. Around these disks, in the shape of a tube, is formed the copper outer conductor. For physical strength and additional electrical shielding, two layers of steel tape are helically wrapped around the whole unit. This is the 0.375 in. diam longitudinal seam-type of coaxial unit and is illustrated in fig. 1.

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A critical factor in the transmission characteristics of a coaxial circuit is a variation in its electrical impedance. These points of impedance variation along a transmission line cause signal reflections or echoes and consequent ghosts in television pictures. These effects can be corrected partially along the line but excessive impedance deviation within the cable would necessitate costly equalization equipment. The actual value of impedance in a coaxial unit is determined by the diameter of the center conductor, the diameter of the outer conductor, and the insulation employed between them. Thus, any variation in dimensions will cause a variation in impedance with its consequent signal reflection. This impedance variation is critical enough to require manufacturing dimensional tolerances of ± 0.0001 in. on the diameter of the center wire and ± 0.0002 in. on the thickness of the outer tube to obtain the necessary uniformity of tube diameter.

The signal quality is affected by another electrical phenomenon known as corona, which may occur because of the use of high voltage for transmitting power to operate repeater stations along the cable. This effect causes noise interference with the signal and must not become appreciable. A number of causes contribute to this corona effect, not the least of which is the condition of the conductor surfaces. Any slight irregularities such as pits, flakes, scores or slivers can contribute to this effect and also increase the possibility of an arc or breakdown between conductors in the unit. The electrical characteristic requirements and the physical requirement

Fig. 1—This single coaxial unit (above) with its outer conductor opened shows the construction. The entire assembly is fabricated in the forming machine.

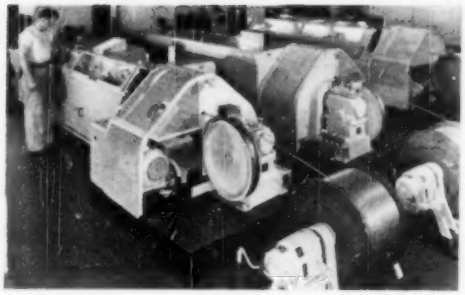

Fig. 2—The coaxial forming machine, (left) is viewed from the takeup end. Forming operations take place in full view of the operator, and up to 1700 ft of unit may be wound on the takeup reel.

of flexibility posed difficult problems in commercial mass production of coaxial units.

The unit is assembled in one machine. This machine magazines, slits and spaces the disk insulators on the center conductor; notches and embosses the edges of the copper tape for interlocking the edges after forming into a tube; and forms the tube around the disk insulators by drawing the entire assembly through a forming die. The machine then helically wraps two steel tapes over the copper tube, and takes up the unit on reels for later processing into cables. The assembly is one continuous operation. Fig. 2 shows one of the forming machines from the head or takeup end. All forming operations may be viewed through glass windows.

Prior to assembly the insulating disks are punched from strips of polyethylene and supplied to the hopper at the forming machine. The center conductor, copper tape and steel tape are prepared for the forming machines by preliminary operations.

The condition of the conductor surfaces has an influence on the ability of the unit to withstand the high potential difference necessary for power supply and at the same time maintain a negligible level of undesirable noise. To assure a smooth clean conductor surface, the copper rod from which the center wire is drawn is shaved to remove scale, pits and other surface imperfections that might carry through to the finished wire.

The dimensional requirements of the coaxial unit, along with the method employed to form the tube continuously around the unit, demand consistency in physical dimensions, electrical conductivity and workability of the copper tape. As the highest conductivity is desired of the outer tube, electrolytic tough pitch or oxygen-free high-conductivity copper is used. To obtain maximum workability of the tape, it is annealed dead soft before use in the forming machines.

Surface scale, slivers and other imperfections of the tape surface have been most persistent problems. The surface scale, usually invisible while the copper is in tape form, appears after the tape is worked or formed into the tube.

The extremely low dielectric losses of polyethylene used in insulating disks, at frequencies of several megacycles, its relative chemical inertness, its resistance to moisture and its physical toughness made it a natural choice.

The steel tape used as the outer covering for the unit receives no processing other than rewinding to spools designed especially for use in the forming machines.

The production of coaxial unit is performed on 40 forming machines, requiring one operator for each pair of machines. Western Electric Co.'s Point Breeze plant manufactures all the coaxial unit for the Bell System. The unit is made to specified lengths, the maximum length on any one reel being about 1700 ft.

After the coaxial unit is formed and tested it is stranded with various paper insulated wire circuits into a cable. This stranding is performed by wrapping the four, six or eight units helically around a core of wire circuits. Other wire circuits may also be wrapped over the coaxial units in layers, each layer being wrapped in an opposite direction to give flexibility to the cable. This cable core, as it is then called, is dried, employing heat and vacuum to remove moisture from the core. After the core is thoroughly dried, a protective covering of lead is applied. Fig. 3 illustrates a cut-away view of a typical coaxial cable in finished form.

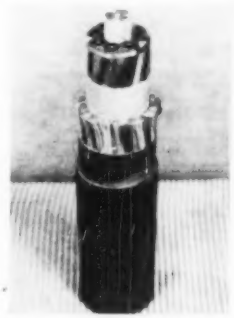

Fig. 3—Construction of a typical 8-unit coaxial cable is shown in this cutaway section. Cables are made up in four, six, and eight units.

Over the lead sheathing there may be an armor cover applied. This depends upon the locality and ground condition where the cable is to be installed. After armoring, the cable is subjected to exhaustive electrical tests, filled with nitrogen under pressure and sealed to exclude moisture.

The reels of finished coaxial cables are spliced together on installation to form a continuous unit between repeater stations, which at present are about eight miles apart. Each splice, each coaxial terminal and each repeater offers points where manufacturing deviations can cause impedance deviations with consequent signal degradation. Since many such points are in tandem in a long circuit, excessive variations in the cable would make the necessary remedial measures, in the form of gain and phase equalization, more exacting and costly.

The Calbeck Process for Refining Zinc Oxide

by John H. Calbeck,
William T. Maidens
and Oscar J. Hassel

The rotary gas fired reheating furnace used by the American Zinc Oxide Co. at Columbus, Ohio for refining lead-free zinc oxide is described. The outstanding features of this operation are that the color of the zinc oxide is greatly improved, sulphur is eliminated, and cadmium recovered without densifying the product to an objectionable degree.

IN 1919 Leland S. Wemple obtained a patent for a process of reheating zinc oxide wherein the "coarsening of grain due to excessive heating was avoided." He taught in his specification that if solid carbonaceous material, such as lamp black, was added to the zinc oxide in proper amounts prior to reheating, objectionable sulphur compounds could be removed and the color would accordingly be improved and no objectionable densification would occur because of the relatively low temperature required. The situation that made this invention imperative was the newly opened zinc oxide plant of the American Zinc, Lead & Smelting Co. in Hillsboro, Ill. This was one of the early Western Type American Process zinc oxide operations. Characteristic of all of these early Western operations using Tri-State and Western ores was the great difficulty encountered in obtaining a product low enough in sulphur to compete with the Eastern Type American Process zinc oxides which were made from ores containing very low sulphur percentages. Wemple demonstrated that the refining process of his invention produced a superior color and although this was true and a most welcome feature, the primary purpose of the early refining operations at Hillsboro was to reduce substantially the high sulphur content of the crude zinc oxide.

Although many and varied attempts had been made for refining zinc oxide none of the processes had a commercial history of any consequence until Wemple's invention became standard practice for the American Zinc, Lead & Smelting Co. in 1919 and their operations have been unique in that substantially all of their lead-free zinc oxide has been reheated since the first installation at Hillsboro. This process has become known in the industry as refining.

The furnace developed by Wemple and continued in use by the company from 1919 until 1943 was unusual and merits some consideration by way of review in this paper. The furnace was essentially a double hearth coal-fired muffle furnace with a mechanical raking system consisting of a central shaft supporting six rabble arms in each muffle. The untreated or "crude" zinc oxide was fed onto the outer rim of the top muffle, moved to the center where it dropped to the lower muffle and progressed to the outer rim where it was discharged into an alloy screw conveyor. The retention in this furnace was extremely short, about 5 min, and the shallow zinc oxide bed on the hearths of the muffles was being continuously turned by the fast moving rabbles. Soft coal was burned on the grates below the lower muffle and the long yellow flame necessary to carry the heat around both muffles resulted in a very inefficient combustion of the fuel. The temperature of the top of the lower muffle seldom exceeded 650°C although the oxide itself often reached 700°C before discharge. The capacity of this furnace was approximately ½ ton per hr. In our plant at Columbus it was necessary to keep four of these furnaces running in parallel to take care of the production because, as mentioned above, every pound of zinc oxide produced during these 24 yr passed through one of these refining furnaces. An essential part of this refining operation was the use of carbonaceous material admixed with the zinc oxide fed to the furnaces. Between 1 and 2 pct of a bran produced in the processing of cotton seed was added to all zinc oxide charged to the furnaces. The bran ignited on the top hearth and was still burning when the charge fell from the top hearth to the bottom hearth making a cascade of sparks. The rapid turning of the zinc oxide caused these particles of bran to flash on the hearths behind each rabble; but the combustion, of necessity, had to be complete by the time the charge reached the outer rim of the bottom hearth, otherwise the finished product would be contaminated with the charred particles of bran which would give the zinc oxide an unsatisfactory color. Although this operation was initiated to reduce objectionable sulphur percentages, as time went on new properties of the product were appreciated which made advisable continuing the refining process long after other methods of sulphur reduction became known in the industry. The particle size and particle size distribution, the absence of colloidal fines and perhaps a unique surface condition gave this product an outstanding performance when used in paints.

The Wemple furnaces installed in Columbus in 1919 had to be rebuilt frequently and were extravagant in the use of fuel. The raking mechanism and the muffles required excessive maintenance expense and as the furnaces wore out the problem arose whether to continue along this line or to explore the possibilities of obtaining similar or better results in the simpler and more commonly used rotary furnace. To this end special research was initiated in 1941 on a small laboratory rotary

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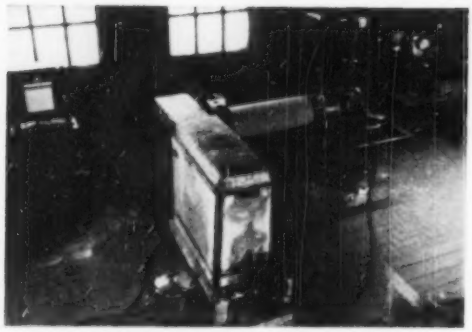

Fig. 1—Experimental refinery kiln including large combustion chamber or "Dutch Oven."

furnace illustrated in fig. 1. Studies with this little apparatus developed all of the qualities of the Wemple furnace and in addition it was discovered that between 66 and 90 pct of the cadmium content of the crude oxide could be recovered. The rapidly growing demand for cadmium made this a most attractive feature. A large rotary furnace was installed in 1942 and has been in continuous operation at Columbus since 1943.

Equipment of the Process

The furnace installation that emerged after more than a year of tedious development, work and study is illustrated in fig. 2. The installation consists of three essential parts: 1. The rotary furnace proper with the usual drives feeding and discharging equipment. 2. The dust collection equipment. 3. The gas proportioners, combustion chamber, automatic controls, recirculation system, and the arrangement for periodically alternating the furnace atmosphere from reducing to oxidizing.

The Furnace: The furnace consists of a rotary kiln the shell of which is 6 ft id and 70 ft long mounted in the usual manner with the exception that a slope of 1 ipf is used rather than the conventional ½ ipf. Special air-tight seals were designed for both ends. Fig. 3 is a drawing of the seal on the discharge end of the kiln and fig. 10 is a photograph of the furnace showing the seal. This type of seal has proved exceptionally efficient. As will be seen from the drawing, the packing gland is designed to function only in a direction parallel to the axis of the furnace and is not affected by the rotary motion which is most destructive to the packing in a gland of this diameter. The rotary motion is taken care of by the wearing ring and plate which provide also for whatever eccentricity that may develop.

The furnace is insulated and lined throughout its length with 2 in. of Johns-Manville insulation and inside of the insulation with 4 in. fire clay kiln blocks for 64 ft of its length. At the discharge end, 6 ft is lined with Carbofrax circle brick. This was found necessary to prevent incrustations forming too rapidly. The discharge end requires occasional cleaning with a cutting tool.

The crude oxide from the baghouse is delivered by a belt conveyor to a 30 ft bucket elevator and then to screw conveyors which convey it to one of the two rotary refining furnaces. The screw conveyors delivering the oxide to the furnaces are equipped with seals to prevent the cadmium laden fume of the furnace from escaping to the building through the conveyors.

The crude zinc oxide is fed at a uniform rate and because of the slope moves rapidly through the first 20 ft of furnace length. As a matter of fact, it moves so rapidly that lifters or retarders have to be provided to hold back this rapid movement through the remaining 50 ft or reaction zone. Fig. 4 and 5 are photographs of the interior of the furnace showing the retarders. The movement of the oxide through the furnace is more rapid than the normal equation for the movement of solids through rotary kilns would indicate. When zinc oxide has reached a red heat the agitation due to rotation (1.88 rpm) is such as to fluidize partially the charge and if it were not for the retarders the oxide would run through the furnace like water reducing the retention time after red heat to less than 2 min. The retarders, there-

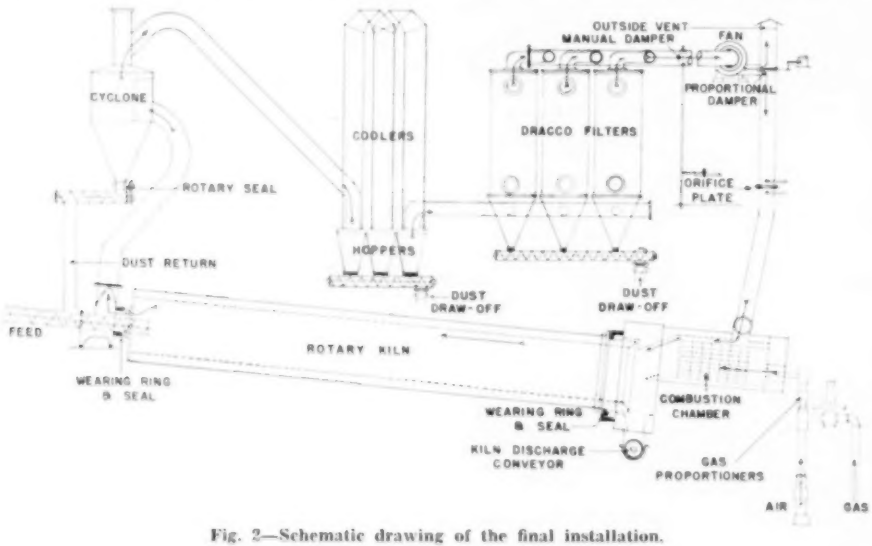

Fig. 2—Schematic drawing of the final installation.

fore, inhibit this rapid flow and retain above each retarder a pool of partially fluidized zinc oxide. The net result is a retention time of between 15 and 35 min depending upon the density of the crude oxide being fed. The retarders also provide ample agitation in the reaction zone. There are 5 rings of retarders spaced 10 ft apart in the reaction zone. Each ring consists of 8 castings of an alloy analyzing 10-13 pct nickel and 26-28 pct chromium.

The discharge screw is of cast alloy flights mounted on a 4 in. stainless steel tube. The hot oxide is dropped on a steel apron conveyor 36 in. wide and 80 ft long running at a rate of 20 fpm which delivers the oxide to a 100 ft rubber belt conveyor which conveys it to the packing house. The discharge screw and the end of the apron conveyor are shown in fig. 10. Occasional stationary rakes along the length of the apron conveyor turn the oxide and level it so that it has cooled to a safe temperature by the time it reaches the belt conveyor.

Dust Collecting System

The gases issuing from the furnace are laden with a mixture of cadmium fume and zinc oxide dust and all of this must be recovered. The gases first pass through the cyclone where most of the zinc oxide dust falls out. (Fig. 2.) The cadmium content of this dust will run from 2 to 10 pct but it is all returned to the feed and represents approximately 1.3 pct of our daily production. This is a remarkable figure in view of the agitation caused by the lifters, the speed and slope of the furnace, and the light and dusty character of the product being treated. Additional dust and fume pass from the cyclone to the coolers from which about 600 lb per day is recovered. This production will run approximately 17 to 20 pct cadmium and is too valuable to return to the feed. An additional 700 lb of dust is recovered from the three Dracco filters every day and has a cadmium content of between 22 to 40 pct. The dust from the coolers and the filters is combined, packed in paper bags and is either sold or treated for the recovery of metallic cadmium at our Fairmont City plant.

This Dracco filter installation merits some special discussion. This unit consists of three tanks in parallel and each contains 18 woolen bags 8 in. in diam at the bottom, 5 in. in diam at the top and 117 in. long. The standard Dracco compressed air shaking mechanism is used but the rotary Dracco timing valve has been replaced by three solenoid valves (one for each tank) which are actuated by the electrical timing device that times the alternations of the burners to be described later. The shaking is thereby synchronized with the burners so that shaking always occurs on an oxidizing cycle when the quantity of solids in the fume is small. Dust collection is heavier on the reducing cycle and maximum efficiency of the bags is required during this cycle. The bags in each tank are shaken for 20 sec during each oxidizing cycle, one tank at a time.

Insulation of the Tanks: All three filter tanks are heavily insulated as are both manifolds, the discharge hoppers, screw conveyor and rotary seal. This is extremely important because the use of woolen bags is possible only if the temperature is maintained between the dew point of the gases and the maximum temperature that the bags can withstand, approximately between 140 and 220°F. The dust collected carries from 2 to 8 pct Cl, as zinc chloride and although this is not destructive to wool at temperatures above the dew point, it is extremely corrosive at lower temperatures. If condensation occurs in the system and water gets on the bags, they are ruined in a few hours. The heavy moisture content of the gases by reason of burning natural gas makes necessary this insulation to prevent condensation at any time in the filter tanks.

To maintain the temperature in the filter tank summer and winter, the cooling tubes are enclosed in a housing with adjustable louvers at the top. Manual control of these louvers regulates the circulation of air through the housing and controls the temperature satisfactorily. The common expedient of allowing diluting air to enter the gas stream to control the temperature cannot be resorted to because a tight system must be maintained and no tramp air admitted.

Burners, Combustion Chamber, Automatic Controls and Recirculation Feature

Natural gas is used for fuel and the burner setup is illustrated in fig. 6. Two proportional mixers are

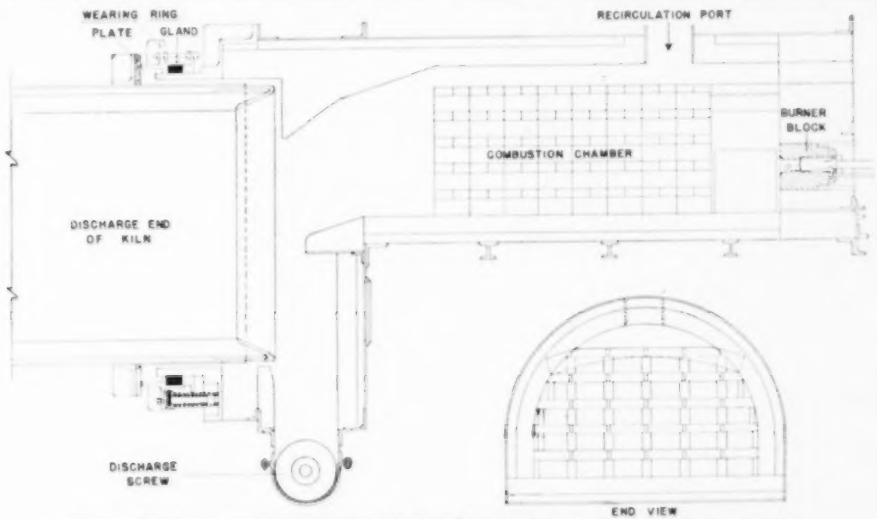

Fig. 3—Discharge end of furnace including "Sealing Ring" and combustion chamber.

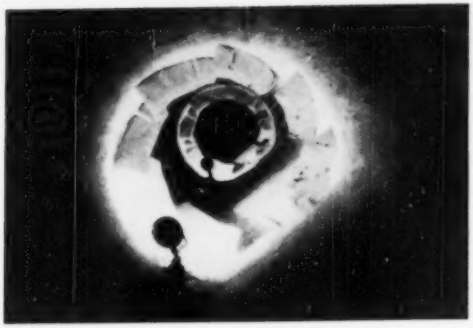

Fig. 4 and 5—Lifters or retarders.

mounted in parallel and a damper system operated by an electrical timer turns off one and turns on the other simultaneously at regular intervals. The setting of one proportioner is extremely lean, usually with about 75 pct excess air and the other one is correspondingly rich, usually analyzing about 50 pct deficiency in air. Now what happens is this: when the furnace is heated and in normal operation the oxidizing flame runs for 4 min. This flame passes through the checkerwork of the combustion chamber (fig. 2), and part of the heat is absorbed by the checkerwork and the remaining partially cooled gases pass into the furnace proper. When the 4 min period elapses, the electrical timer switches the dampers so that the reducing proportioner is put in operation and the oxidizing proportioner is turned off. This goes on for 5 min. When this overrich mixture strikes the hot checkerwork of the combustion chamber, a reforming of the excess methane results so that the composition of the gases entering the furnace are extremely reducing as shown in table I. These gases, surprising as it may seem, are substantially clean and free of carbon or soot and may contain as much as 20 pct of hydrogen and carbon monoxide. In other words, one-fifth of the gas entering the furnace on the reducing cycle may be carbon monoxide and hydrogen. This reforming action, being endothermic, cools the checkerwork of the combustion chamber but the heat is replaced on the next oxidizing cycle. It is obvious that a careful adjustment must be made so that the heat withdrawn on the reducing cycle is replaced on the oxidizing cycle. If too little heat is applied on the oxidizing cycle, the checkerwork cools down to the point where the reducing flame becomes smoky. If too much is applied, the temperature of the checkerwork increases to the softening point of the packing and its life is shortened needlessly.

Two L & N Micromax recorder-controllers are used for temperature control. One is connected to a chromel-alumel couple at the discharge end and the other to an iron constantan couple at the feed end. The one at the feed end is located in the gas stream but the one at the discharge end is placed where the oxide as it is discharged from the furnace cascades over the alloy protecting tube of the thermocouple. This gives an accurate temperature of the oxide when the furnace is discharging hot oxide but is very inaccurate when no oxide is passing because it is located too low to register the temperature of the gases.

These L & N controllers operate separate butterfly valves in the blast line through Barber-Coleman motor controls (fig. 6). The two proportioners are of the type that utilize the entrainment principle of *a venturi* to obtain a precise single valve control of the gas-air mixture ratios and therefore complete control of the burners is obtained by the butterfly valves in the blast line. There is also a butterfly valve for manual control and these three regulate the blast pressure in the manifold directly below the proportioners. There is a butterfly valve for each proportioner above the manifold. These are for alternating the mixtures. When one is on, the other is off and these are operated by the Barber-Coleman motor which in turn is actuated by the electric timer that controls the cycling. In fig. 7, a control panel for No. 1 furnace is shown. To the left of center is a weightometer control and a rateograph recorder. At top center is a Hays 5-unit draft indicator. At the top left is the Draftrol which controls the proportioning valve. At the top right is a Hays recorder which records the burner manifold pressure and the recirculation differential set up in the orifice plate. The two Micromax controller-recorders are on the right below the Hays' instruments. The timers are shown at the lower left.

All of the zinc oxide produced at our plant is collected in a single baghouse and delivered to the refinery on a belt conveyor. A weightometer is installed as part of this conveyor system which automatically controls the operation of the collecting screws so that the oxide is delivered to the belt at a fairly uniform rate. A recorder provides a rateograph of the feed to the furnace which runs between 4000 and 6000 lb per hr.

A most important part of the unit is the recirculation of part of the filtered gases through the furnace. A variable fraction of the spent filtered gases are recirculated to provide a constant velocity of the gases over the charge. This is controlled automatically by the proportional damper (fig. 2). This damper shunts a fraction of the filtered gases back into the combustion chamber to mix with the combustion gases and the remainder is exhausted to the air. The damper is operated by a Barber-Coleman motor control which in turn is operated by the Hays Draftrol adjusted to maintain a zero draft at the discharge end of the furnace. Zero draft is necessary here because there is no seal on the discharge screw that removes the hot oxide from the furnace.

The volume of recirculated gases is measured by the calibrated orifice plate in the line from the proportioning damper to the combustion chamber. The differential of the orifice plate is measured on the Hays Draft Indicator and Recorder in the control

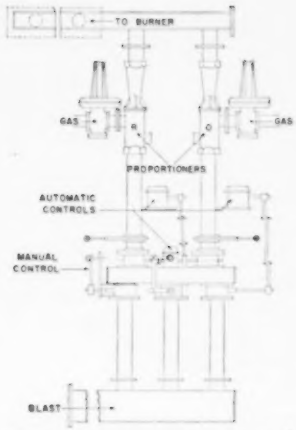

Fig. 6—Gas proportioners and controls.

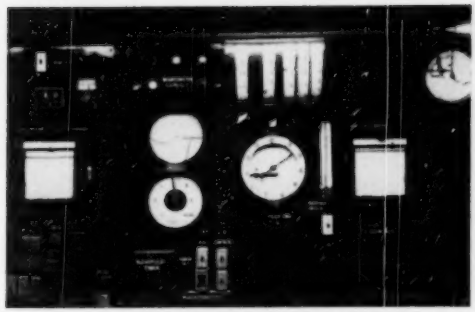

Fig. 7—Control panel.

room. The measured volume of recirculated gases amounts to between 20,000 and 40,000 cfhr. The gas mixture input averages between 15,000 and 30,000 cfhr. That is to say, the recirculated gases dilute the combustion gases on an average of one to one. A high burner turn-up of 30,000 cfhr would be diluted with approximately 20,000 cf recirculated gases and a low turn-down of 15,000 cf would be diluted with 40,000 cf, that is, the dilution ratio varies from 0.6 to 2.6.

The Refining Process

As mentioned above, the oxide progresses rapidly through the first 20 ft of the furnace because of the slope and the rapid rotation (1.88 rpm). No bran or solid carbonaceous material is added to the crude zinc oxide as was done for the Wemple furnace.

Upon reaching the first ring of retarders, the downward progress is reduced and a pool of oxide 10 in. deep forms. Escaping through the spaces between retarders, the oxide hurries on to the second ring of retarders. At this point the oxide has reached refining temperature. Prior to this the oxide has been brought up to this temperature by the heat of the effluent gases. During the first 20 ft, much of the dust in the gas stream settles out because of the slowing down of the gases. There is little agitation of the oxide itself in the first 20 ft, rather it scoots along without much tumbling or cascading.

During the remaining 50 ft of passage through the tube the refining action proceeds. This is in two stages or cycles and consists of 4 min of oxidization and 5 min of reducing. The retention in the reaction zone varies from 9 to 35 min depending upon the density of the oxide. When the lighter acicular oxides (18 to 20 lb per cu ft) are being refined, they pass through the refining zone in slightly more than 9 min, that is, they are subjected to only one complete cycle. When the heavier round particle oxides (30 to 40 lb per cu ft) are being refined, they progress more slowly and are subjected to about two complete cycles in passing through the reaction zone. This refining is controlled as to temperature, composition of the gases and velocity of the gases, and the importance of the latter commands special emphasis.

The refining action requires a short period of contact with both reducing and oxidizing gases but neither is effective unless the velocity of gases passing through the furnace exceeds a minimum of about 20 fps (calculated to standard temperature and pressure), but the gases of combustion necessary to maintain the required temperature do not provide more than one-half that velocity. To turn in more gases from the burners simply increases the operating temperature of the furnace to a point where densification is rapid and cannot be prevented regardless of the velocity or composition of the gases. This difficulty is overcome by diluting the combustion gases with clean recirculated flue gases before they enter the reaction zone of the furnace. The manner in which these diluting gases are introduced and controlled has been described above.

Furthermore, contamination of the recirculated gas with air is no problem on the oxidizing cycle because an excess of oxygen in the gas is required. On the reducing cycle, however, precautions must be taken to prevent oxygen from contaminating the recirculated gases because it causes a partial or sometimes a complete destruction of the reducing atmosphere. Therefore, the entire recirculation system, including the cyclone, cooler, goosenecks and hoppers, filter tanks, fan, proportioning damper and all connecting pipes must be kept air-tight. The system is under a negative pressure of from 0.0 to 6.5 in. W.C. except for the short length from the fan to the combustion chamber which is under positive pressure.

The volume of the recirculation system and the kiln is about 1500 cu ft and at normal operating conditions is completely changed in 1¾ min. Consider the shift from oxidizing to reducing. One and three-fourths minutes must elapse before the recirculated gases entering the combustion chamber are substantially free of oxygen which is present to 8 pct during the oxidizing cycle. This actually cuts down the reducing time of the cycle and is partially compensated by making the reducing cycle longer than the oxidizing cycle. To further offset the oxygen contamination a bleeder valve turns in an oversupply of gas into the reducing mixture for about 1 min at the beginning of each reducing cycle. By the time the reducing cycle is completed (5 min) the oxygen content of the recirculating gases should be less than 1.0 pct by volume. An oxygen content higher than this indicates a leak in the system and must be corrected.

As mentioned above, the purpose of the recirculation system is to maintain gas velocities in the furnace above a critical minimum. This minimum must be exceeded regardless of the "turn up" of the burners. When the burners are turned low (because of manipulation of the automatic heat controls) the volume of recirculated gases returned to the kiln must be high and the volume exhausted to the air

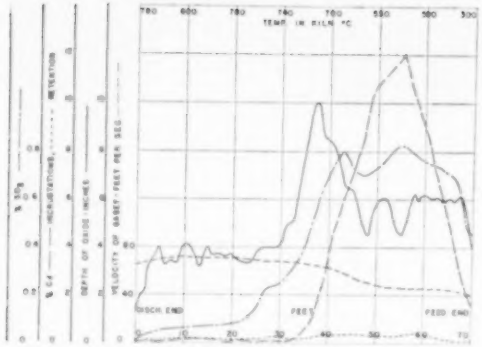

Fig. 8—Graph illustrating conditions on the interior of the furnace.

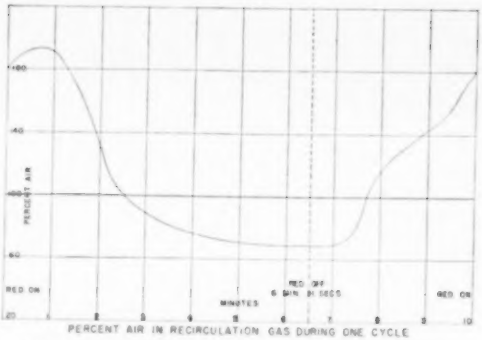

Fig. 9—Graph illustrating composition of recirculated gas throughout a cycle.

must be small; in fact, the latter must equal exactly the burner input to maintain a proper draft balance in the tight system. When the burners are turned up the reverse is true. All this is handled by the Draftrol controlled proportioning damper by simply maintaining 0.0 draft at the discharge end at all times.

For example, suppose the furnace has been operating under light load, the burners are down and the recirculation is up. A surge of cold oxide is delivered at the feed end. The feed end control responds to the sudden drop in temperature and the burners are turned half on. This increase in gas input into the system gives a plus draft at the discharge end, the Draftrol immediately opens the proportioning damper to vent the extra volume and restore the draft to 0.0. Theoretically, the critical gas velocity is that velocity at which cadmium will vaporize in the reducing atmosphere and temperature of the furnace. Actually, the oxide reaches 770°C, the boiling point of cadmium, just before it is discharged but practically all of the cadmium elimination has to be done at temperatures below the boiling point. Furthermore, the high gas velocity lengthens the reactive zone in the furnace and also inhibits densification, the reason for which is not known to us. However, we do know that if recirculation is impaired and the reducing gas percentage falls below a critical minimum, sudden jumps in apparent density of the zinc oxide from 18 to 35 lb per cu ft may occur. The normal increase in apparent density of the packed product over the crude oxide seldom exceeds 1 lb per cu ft or 5 pct.

The recirculation control maintains constant gas velocities throughout the length of the furnace. Calculated to standard conditions this is about 20 fps. Actually, it varies with the temperature and fig. 8 illustrates how this varies with location in the furnace. A high velocity of about 80 fps extends through the reaction zone but drops to about one half of this figure for the feed end beyond the last retarder. Fig. 8 also illustrates the cadmium content of the oxide coating or incrustation of the furnace, the sudden break at 32 ft indicates that above this point cadmium is actually plating out on the incoming oxide during the heating period. In other words, if the cadmium content of the crude runs 0.45 pct then refining begins near the point where the incrustations have that cadmium content.

Second in importance to velocity is maintenance of proper atmospheres in the furnace. Fig. 9 shows graphically the changes that occur during a complete cycle based on theoretical air for perfect combustion, that is, 100 pct is a perfect mixture, below 100 pct indicates a deficiency and above an excess. The speed at which the atmosphere becomes strongly reducing is affected by the amount of raw gas bled into the system at the changeover. Auxiliary air goes in at the change from reducing to oxidizing but the slope is gradual because the auxiliary air stays on throughout the oxidizing cycle.

The composition of the gas delivered by the two burners is checked frequently by a complete gas analysis but a visual check may be made on any reducing cycle by allowing a small jet of recirculation gases to be ignited and burn in a special burner. This is known as the "telltale."

The gas proportioners seldom need adjusting. If the main gas regulator delivers gas to the zero regulators at constant pressure (4 in. WC.), the gas samples analyzed check closely from week to week so that a failure of the "telltale" to "show" usually indicates that an air leak has developed in the recirculation system and not a change in proportioner adjustment. An oxygen content of 1.0 pct or above in the recirculating gases affects the refining quality of the gases. Table I shows typical gas analyses at the peak of the oxidizing and reducing curves.

The chemistry of the reactions in the furnace are well known. Cadmium oxides may be reduced and vaporized at temperatures well below where zinc oxide is reduced and this makes the separation simple. However, some zinc oxide is reduced in our furnace and goes out with the cadmium fume in amounts approximately equal to that of the cadmium. In addition to the true zinc oxide fume there is zinc oxide dust most of which is separated from the fume in the cyclone and returned to the feed. Cadmium elimination may run as high as 85 pct but the average is 65 pct.

Sulphur elimination on the other hand is better. A crude oxide running 0.45 pct sulphur will be reduced to 0.02 pct sulphur or an elimination of 95 pct. When a special effort is made to reduce the zinc sulphate content of the zinc oxide, a product is made that has the fast curing properties of French Process zinc oxide when used in rubber compounding. This is not only caused by the low percentage of sulphur remaining but also by the fact that part of the residual sulphur is present as zinc sulphide. The sulphur is present in the crude mostly as $ZnSO_3$ and with a small amount of $ZnSO_4$. During the reducing

Table I. Composition of Typical Furnace Atmosphere

	Percent by Volume	
	Oxidizing	Reducing
Nitrogen (by difference)	78.8	64.4
Oxygen	6.0	None
Water Vapor	7.6	12.7
Carbon Dioxide	7.6	5.6
Carbon Monoxide	None	7.3
Hydrogen	None	9.9
CH_4 and other Hydrocarbons	None	0.1
	100.0	100.0
Air in Mixture, Percentage of Theoretical for Perfect Combustion	137.5	58.5

cycle the $ZnSO_4$ is largely reduced to $ZnSO_3$ and possibly a very small amount of ZnS. Zinc sulphate decomposes at 740°C but this temperature is barely exceeded at the discharge end. For this reason we feel that the reducing atmosphere reduces the zinc sulphate to zinc sulphite which breaks down readily at lower temperatures. This decomposition occurs during both the reducing and oxidizing cycles. A small amount of ZnS may remain in the calcine. Very little lead is present in the crude and only a small part of it is eliminated (about 25 pct) and this on the oxidizing cycle.

From a chemical standpoint alone, there is little need for the cycling procedure. If high eliminations of cadmium should be required, the oxidizing cycle assists in that the small percentages of cadmium sulphide are reoxidized and made amenable to subsequent reduction but the primary purpose of the oxidizing cycle is to improve the color of the product. This involves not only a high brightness but also a neutral tone, that is, absence of yellow. The yellow is caused by cadmium, lead and trace elements that have not been identified. The brightness is obtained by burning out traces of carbon.

The oxides are thus reheated without substantial densification if proper temperature, composition and velocities of the gases are maintained. There is, however, an incipient sintering action in that the product discharged from the kiln is denser compared with the crude oxide but this densification is eliminated in the subsequent milling step, a step, incidentally, that has always been a part of our refining process.

The milling is now done on two No. 4 Micro mills running in parallel. From the mills the product passes through 6 reels of 46 mesh wire cloth and from there to the packers.

The four Wemple coal-fired furnaces were replaced by the first rotary unit in 1943. A second duplicate rotary furnace was installed in 1949 as a standby. This duplication is necessary because all of the oxide produced must be refined. Failure of a unit does not interrupt the production of the plant. Either unit has ample capacity to refine the maximum capacity of our plant, 120,000 lb per day.

The fuel requirement is between 70,000 and 90,000 cu ft of natural gas per 24 hr day. This amounts to about 1600 cu ft per ton of zinc oxide refined. In 1947 the unit refined 17,511 tons of zinc oxide. The marketable cadmium dust produced was 292,514 lb and had an average cadmium content of 22.52 pct, that is, 65,870 lb of cadmium was recovered. The gross weight of the nonreturnable dust produced by the rotary furnace was therefore 0.83 pct of the production. The total dust produced (exclusive of cyclone) was 399,079 lb or 1.13 pct of the production. The cadmium content of the crude oxide fed to the refinery averages 0.40 pct and the cadmium content of the refined oxides is normally reduced to less than 0.1 pct cadmium. This represents an elimination of 75 pct. It must be emphasized, however, that the refining operation is carried out primarily to improve color and eliminate sulphur and that the cadmium elimination is of secondary importance except in a very few cases where low cadmium zinc oxide is required by a rubber manufacturer. The particle size is not affected by refining insofar as microscopic examination can determine but the low reactivity of the refined oxide when used in paint indicates that there has been a substantial reduction in submicroscopic fines.

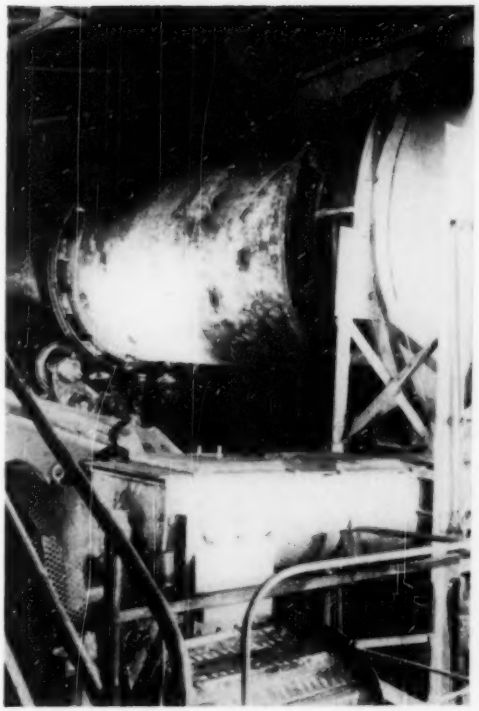

Fig. 10—Refinery interior showing sealing ring, discharge screw and apron conveyor.

The unit may be employed to treat cadmium dusts to recover zinc oxides as a calcine and the cadmium in a concentrated dust. In May 1946, 61,350 lb of low cadmium dust accumulated from refining operations was treated in the rotary unit. The dust had an original cadmium content of 7.29 pct and a zinc content of 66.1 pct. Cadmium dusts as high as 61.5 pct were made and the average for the entire run was 46.7 pct. The zinc oxide residue or calcine averaged 0.67 pct cadmium. The latter was not a marketable product but was returned to the zinc oxide furnace.

U. S. Pat. No. 2,416,044 for the above described process was issued Feb. 18, 1947.

Treatment of Electrolytic Copper Refinery Slimes

by J. H. Schloen and E. M. Elkin

All known methods of treating and recovering the various components of copper refinery slimes are discussed. The slimes treatment processes presently used by five copper refineries are described and flowsheets are given. A bibliography is appended.

THE recent increase in anode copper production at the Noranda smelter placed an additional load on the Montreal East plant of Canadian Copper Refiners Limited. The consequent increase in the amount of anode slimes led to a careful consideration of the various methods of slimes treatment.

A critical review of these methods is presented here in the belief that it would be of interest to copper refiners. No such review has yet been published, although a number of individual flowsheets have been described.

To make the information of greater value, a questionnaire was sent to all copper refineries. While the overall response to the questionnaire was not entirely satisfactory, the foreign refiners gave much detail in their answers and expressed willingness to cooperate further if needed.

Although the data below do not cover all refineries, it is thought that examples of all flowsheets currently in use are included. A comparison of these flowsheets should prove of interest.

The paper is divided into two parts. One part deals with a discussion of all known methods of treating and recovering the various components of the slimes. The second describes the processes now in use at various copper refineries for the treatment of raw slimes up to the production of Doré bullion. A brief description of the two main electrolytic methods of parting Doré bullion is appended.

Since the first commercial application of electrolytic refining of copper by J. B. Elkington in 1865, the treatment of anode mud or slimes has undergone many changes.

The object of treating slimes has always been the recovery and separation of precious metals. With the passage of time, increasing attention has been paid to improvement in recovery and quality of by-products, mainly selenium and tellurium.

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The original methods of treatment were crude, although relatively direct. For example,[*] the slimes were fused on a dolomitic clay bed in a reverberatory furnace; the slag was removed and the resulting metal cupelled to silver and gold. In another process, the dried slimes were oxidized and scorified on the lead bath of a cupelling furnace. Later, these pyrometallurgical methods were modified by sulphuric acid leaching, followed by fusion with niter in a reverberatory furnace. A preliminary oxidizing roast improved the leaching step. Straight hydrometallurgy has not found practical application.

No matter what method is employed, the result is Doré metal—a silver-gold alloy, with or without the metals of the platinum group.

Characteristics of the Slimes

Anode slimes are made up of those components of the anodes which are not soluble in the electrolyte. They contain varying quantities of copper, silver, gold, sulphur, selenium, tellurium, lead, arsenic, antimony, nickel, iron, silica, etc.

The color of raw slimes is grayish black and the particle size is —200 mesh. However, Baraboshkin and Gaev (Ref. 1, p. 37), describe light gray colored slimes from the electrolysis of secondary copper which are composed predominantly of sulphates and of basic salts. In anode furnace practice primary and secondary metals are not usually segregated and such slimes do not commonly occur.

Copper in the slimes originates to a large extent in the cuprous oxide of the anodes and is therefore dependent on the oxygen content of the latter:

$$Cu_2O + H_2SO_4 \rightarrow CuSO_4 + H_2O + Cu \quad [1]$$

Opinions have been expressed that some of the copper is formed by decomposition of cuprous to cupric sulphate

$$Cu_2SO_4 \rightarrow CuSO_4 + Cu \quad [2]$$

The product of either reaction is very finely divided.

Depending on the slimes composition, much of the copper is combined with sulphur, selenium, tellurium, etc. The balance of the copper is in the free metallic state.

Apart from copper (and the platinum group elements) the only slimes component existing in uncombined state is gold, although it may be partly combined with tellurium and to some extent alloyed with silver.

The chemical stability of slimes is that of the constituents.

At room temperature the slimes are not appreciably oxidized and in the absence of air they are unaffected by dilute sulphuric or hydrochloric acid, though readily attacked by dilute nitric acid. In the presence of air, copper is slowly dissolved by dilute sulphuric acid with the liberation of selenium and tellurium. The rate of the reaction increases with temperature. The liberated tellurium is partly oxidized and dissolved by hot dilute sulphuric acid.

When heated in air the slimes are oxidized with the formation of the corresponding oxides, selenites and tellurites. Selenium and tellurium oxides are also formed and may be partly volatilized.

On heating the slimes with strong sulphuric acid, oxidation and sulphation take place. Copper, silver and base metals are sulphated, gold is unaffected, while selenium and tellurium are oxidized to the corresponding oxides or oxysulphates. On raising the temperature to that of roasting, the oxysulphates decompose and selenium dioxide driven off.

Treatment of Slimes Components

Direct smelting of slimes is no longer practiced. Although a high initial gold fall is obtained, excessive matte and slag formation causes heavy recirculation of precious metals, especially of silver. A much more satisfactory approach is separating the component elements one by one. In this manner they may be recovered in a reasonably pure state permitting either immediate withdrawal from the circuit or recirculation with a reduced number of treatment steps.

The most favored procedure is the removal of copper and then of selenium and tellurium. Other elements, apart from precious metals, are not recovered, except for lead, and that only in the close proximity of a lead smelter. The end product is Doré metal—a silver-gold alloy containing a small quantity of copper and minor amounts of other impurities.

In an ideal process the various impurities would be recovered in a pure state separately, as early and as completely as possible, so as to leave little refining to be done in the Doré or cupelling furnace, with minimum consumption of fuel and fluxes and minimum formation of matte and slags. In practice many compromises must be made, mainly because no element is removed entirely and because more than one element is usually removed at any one time.

Removal of Copper: As a rule, copper is removed as a water soluble sulphate. In this form it may be sent directly into the electrolyte circulation system. However, due to the presence of impurities, such as selenium and tellurium, this is practiced in few instances only. Generally, the leach solution is worked for the recovery of copper sulphate, or, if the impurities content is too high, sent to the liberator system.

In few cases only are the liberator cathodes of commercial quality. They are usually remelted in the anode furnace. The liberator solution may be concentrated although the strong acid is quite impure and usually not suitable for electrolytic work.

Following are the commercial methods for recovering copper:

Oxidizing Roast:[1, 4, 5, 7, 8, 10, 15, 17, 19, 43] Shallow layers of slimes are heated to 500-800°F. Sufficient air is admitted both to effect oxidation and to hold the temperature below the fritting point. The bed is generally not rabbled to avoid dust losses although if efficient dust collection is present, rabbling is employed to reduce the roasting time. Copper compounds are broken down and together with free copper form copper oxide, selenite and tellurite. Some selenium and arsenic are volatilized. The calcine is leached with dilute sulphuric acid, foul electrolyte or liberator solution. The solution is then sent to electrolytic department, copper sulphate recovery or liberator system, as discussed above.

The main objections to roasting are dust losses which introduce recirculation, only partial elimination of the more volatile impurities and an impure leach liquor containing selenium and tellurium. This selenium is precipitated as copper selenide in the liberator system and is not easily recovered in a flowsheet employing this method.

A variation of this method consists of heating slimes mixed with fine soda ash. This will be described under selenium removal.

Sulphatizing Roast:[1, 2, 4, 5, 15, 20, 21, 22, 39, 41, 44] This method is an improvement over air roasting in that concentrated sulphuric acid is added to the slimes prior to charging or during the roast by spraying. The acid acts both as an oxidizing and a sulphatizing agent.

$$Cu + 2H_2SO_4 \rightarrow CuSO_4 + 2H_2O + SO_2 \quad [3]$$

Other metallic components react similarly. Selenium and tellurium are oxidized to the corresponding dioxides or oxysulphates

$$Se + 2H_2SO_4 \rightarrow SeO_2 + 2H_2O + 2SO_2 \quad [4]$$

$$Te + 2H_2SO_4 \rightarrow TeO.SO_4 + 2H_2O + SO_2 \quad [5]$$

Since the acid is required both for oxidation and for sulphation of the copper, its requirement is double that used in methods described under *Oxidizing Roast and Aeration in Dilute Sulphuric Acid.*

Roasting at high temperature drives off the selenium dioxide. To prevent decomposition of copper sulphate, temperature must not exceed 1200°F. Yet at C.C.R., roaster temperatures have reached 1400°F, with no noticeable effect on copper sulphate.

The roasting step may be modified by a preliminary heating of the slimes-acid mixture below the boiling point of the acid.[20, 22, 39, 44] The sulphur dioxide and water given off and the base metal sulphates formed increase the porosity of the mix and aid in retaining the acid needed to complete the oxidation of selenium and tellurium at a higher temperature. Roasting then merely drives off the volatile oxides.

The roasted slimes are leached with hot water. The leach liquor carries much silver and some selenium and tellurium. Silver is readily cemented on metallic copper or raw slimes. Selenium requires a somewhat longer cementation time, while tellurium is more resistant to cementation than selenium. Both require an acid environment.

The reactions involved are complex, but may be represented by the overall equation

$$4Cu + 2H_2SO_4 + H_2SeO_4 \rightarrow 2CuSO_4 + Cu_2Se + 3H_2O \quad [6]$$

As practiced at C.C.R., digestion-sulphatizing-roasting of slimes has given satisfaction with good

Table I. Anode Analyses

	C. C. R.			AUSTRALIA		NORDDEUTSCHE			
	BOLIDENS	DOMESTIC	NORANDA	E.R.&S.	MT. LYELL	I	II	OUTOKUMPU	RHODESIA
Cu pct	99.00	99.56	99.55	99.37	99.21	99.22	97.68	99.04	99.78
Ag oz/t	36.00	53.55	22.20	19.33	3.79	14.9	109.	10.29	1-2
Au oz/t	4.5	3.60	4.35	5.72	0.70	0.57	0.44	0.43	0.01-0.02
Se pct	0.2	0.12	0.15	0.0186	0.021	0.02-0.04	0.175	0.017	0.141
Te		0.03	0.022	0.042	0.0093				0.0019
As	0.004	0.018	0.0006	0.103	0.0019	0.114	0.764	0.008	0.0011
Sb	0.015	0.011	0.0001	0.077	0.0083	0.038	0.113	Tr	0.0002
Bi	0.004			0.0021	Tr	0.003	0.007	0.002	0.0045
Pb	0.05	0.082	0.0026	0.149	0.0073	0.04	0.101	0.0046	
Ni	0.20	0.012	0.019	0.051	0.0195	0.0195	0.35	0.71	0.0154
Co								0.0004	0.0060
Fe	0.003	0.0009	0.0004		0.0422			0.014	0.005
S	0.015	0.0033	0.0024		0.12			0.005	
Sn						0.07	0.07	0.0008	
Zn	0.015				Tr			0.0035	
Insol (SiO_2)	0.003				0.039			0.002	
CaO					0.072				
O	0.25	0.12	0.18					0.18	

Table II. Raw Slimes Analyses

	C. C. R.			AUSTRALIA			
	BOLIDENS	DOMESTIC	NORANDA	E.R.&S.	MT. LYELL	OUTOKUMPU	RHODESIA COPPER REFINERY
Cu pct	40	37.3	45.8	13.77	67.23	11.02	43.55
Ag oz/t	3300	5467	3712	2217	332	2800	1815
Au oz/t	450	382	697	577	61	150	13.4
Se pct	21	20.54	28.42	2.96	3.28	4.33	12.64
Te	1	2.97	3.83	2.58	Tr		1.06
As	0.8	0.57	0.33	4.03	0.70	0.70	0.29
Sb	1.5	0.48	0.31	8.34	0.05	0.04	0.06
Bi	0.8			0.02	0.001		0.48
Pb	10			23.77	1.00	2.62	0.91
Ni	0.5	0.17	0.23	0.49	0.05	45.21	0.27
Co	0.02						0.09
Fe	0.04	0.61	0.40	0.35		0.60	1.42
S	3.5			7.80	9.88	2.32	6.55
						SO_4 2.17	
Sn						1.00	
Zn	< 1					Tr	
SiO_2	0.3			9.72	1.4	2.25	6.93
Al_2O_3				0.71		Tr	1.03
CaO	0.05			0.55			0.54
O	3.0					17.89	
Lb per T	28	19	13	20	15.6	7-8	2.8

Table III. Treated Slimes Analyses (to Doré or Cupel Furnace)

	BOLIDENS	C.C.R.	E.R.&S.	NORDDEUTSCHE	OUTOKUMPU
Cu pct	1.5	3.73	2.18	2-3	0.53
Ag oz/ton	8490	11,900	3367	408-612	10,749
Au oz/ton	1320	1200	897	26-131	564
Se pct	2.0	6.56	0.70	4-10	12.92*
Te		3.38	4.35	0.6-1.3	
As			1.04	3-4	0.31
Sb	4.4		9.60	5-7	0.21
Bi	2.3		0.05	0.4-1	
Pb	30.0		28.79	9-18	11.76
Ni	0.7		0.39	0.6-4.0	0.85
Fe				0.25	0.3
S			5.62		
Sn				1.1-4.0	1.27
SiO_2	1.9			3-9	13.26

* ≦2 leached slimes before H_2SO_4 roast for selenium removal.

removal of selenium, and later ready leaching of copper.

Boiling with Dilute Sulphuric Acid and Niter: The actual reagent is the nitric acid liberated by the action of sulphuric acid on niter. Niter utilization is good, but chemical cost is high. Fume disposal creates a problem. The solution is impure and in addition to copper contains silver, selenium, tellurium, etc. Corrosion of equipment is severe. Introduction of sodium salts into the electrolyte increases the electrical resistance and the specific gravity. Recovery of copper sulphate is adversely affected by the increased volumes of mother liquor formed.

This method is mainly of historical interest (Ref. 23, p. 108).

Aeration in Dilute Sulphuric Acid: In this method the slimes suspended in hot dilute acid are blown with air. Copper may be reduced to a fraction of 1 pct. The process is economical of reagent—one mol acid per atom copper. The saving in acid as also in fuel is partly offset by the cost of steam. Equipment required is simple, though bulky. Corrosion presents no problem. The time required to bring copper to 0.5 pct depends on the initial copper content and on the copper compounds present.

In a patented variation,[44] ferric sulphate is added toward the end of the treatment to hasten dissolution

$$Cu + Fe_2(SO_4)_3 \rightarrow CuSO_4 + 2FeSO_4 \quad [7]$$

$$2FeSO_4 + H_2SO_4 + {}^1/_2O_2 \rightarrow Fe_2(SO_4)_3 + H_2O \quad [8]$$

This addition seems hardly necessary since commercial electrolyte commonly used as the source of acid always contains iron as an impurity.

Any solution containing free acid may be used in this method. Silver is not dissolved but the selenium and tellurium combined with copper are set free. Some of the tellurium is oxidized and dissolved in the hot acid solution making it unsuitable for return to the electrolytic department.

Copper sulphide and nickel oxide are not affected

by acid aeration, and require a more drastic method of treatment.

Pyrometallurgical Methods: As discussed earlier, direct smelting is no longer used. (Ref. 1, p. 38-39; ref. 23, p. 109.)

An attempt was made to smelt slimes in a slag resistance electric furnace. The two products obtained were a copper-lead-silver-gold alloy which could not be easily parted and a copper selenide-copper telluride matte. The process did not prove to be practical.

Smelting, scorifying and cupelling are reserved to the end of the treatment, after most of the impurities have been removed by a combination of roasting and leaching methods.

Addicks, (Ref. 23, p. 123) suggested blowing selenide matte in small converters. Baraboshkin and Gaev, (Ref. 1, p. 54) attempted blowing Cu-Ag-Se matte in the smelting furnace. They found that removal of selenium proceeds slowly, that the matte is avidly saturated with oxygen and that the silver is strongly held in the matte in the presence of cuprous oxide. They concluded that the reactions corresponding to normal converting of copper matte do not take place.

Miscellaneous Methods: A number of methods has been proposed, none of which achieved practical prominence.

Aeration of slimes in an aqueous solution of ammonium hydroxide and ammonium carbonate produces a water soluble cuprammonium complex. This method is costly and offers no material advantage over the similar sulphuric acid method.

The use of manganese dioxide as an oxidant in sulphuric acid solution

$$Cu + MnO_2 + 2H_2SO_4 \rightarrow CuSO_4 + MnSO_4 + 2H_2O \quad [9]$$

introduces an expensive contaminant into the solution. Manganese sulphate is eventually wasted after copper sulphate recovery.

Aeration of slimes in aqueous potassium or sodium cyanide is feasible. The resultant solution contains a number of metals which still have to be separated. That, together with the cost of the reagent, makes the process impracticable.

Passing chlorine through heated slimes removes a number of volatile chlorides. Due to the highly corrosive nature of the hot gases and vapors, this method has never left the laboratory.

Removal of Selenium: Wherever selenium occurs in appreciable quantities, market value makes its recovery attractive. In consequence, the slimes treatment flow sheet must be chosen with a view of greatest and most economic recovery of the element in the purest state possible. Actually, many compromises must be made.

The most important methods of recovering selenium are smelting with soda and niter, roasting with soda and roasting with sulphuric acid.

Soda Smelting:[1, 5, 9, 19, 20, 22] This is a pyrometallurgical method. The raw slimes are first decopperized by any method described earlier. In a few refineries, adjoining a lead smelter, the lead is removed by cupellation. Elsewhere the treated slimes are smelted in a Doré furnace.

The slimes, as charged, are mixed with soda and silica. The first slags are mainly siliceous and contain such impurities as iron, arsenic, antimony and lead. The molten charge is then blown with air to oxidize and volatilize as much as possible of arsenic, selenium and tellurium. The volatilized oxides are caught in the scrubber-Cottrell system. The soda slags which follow are high in selenium and tellurium. The melt is then rabbled with niter to oxidize the copper and make it amenable to slagging.

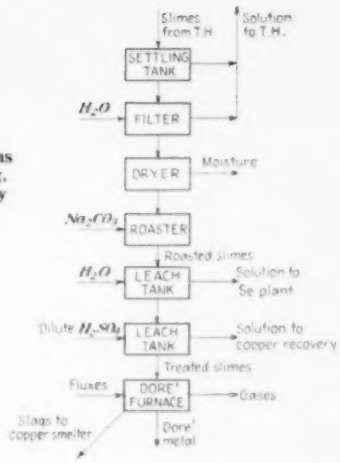

Fig. 1—Bolidens Gruvatiebolag. Silver refinery flowsheet.

The soda slag is crushed and leached with water and the filtered liquor neutralized with fresh sulphuric acid or acid-bearing scrubber solutions to pH 6.2 to precipitate tellurium as tellurous acid. The filtrate is then treated with sulphur dioxide to precipitate selenium. The latter is boiled with water to coagulate colloidal selenium, washed, dried and pulverized. Distillation of the dried selenium markedly improves its purity, but is not commonly practiced.

Selenium recovery is about 80 pct due to inevitable losses in mattes, slags and flue dust.

Replacing soda ash with solid caustic soda improves the efficiency of the process but increases corrosion of the furnace lining.

Scrap iron has been used at C.C.R. in addition to the soda to break down the matte into soda slag and bullion.[5] No blowing was required. Chemical cost and processing time were markedly reduced. Formation of a sodium selenide slag mace selenium recovery more complicated. The process was employed as a temporary expedient only due to the high selenium calcine from the muffle furnaces formerly in use.

Niter treatment tends to oxidize selenium to selenate which is not easily reduced by sulphur dioxide. However, it is needed to break down the copper matte and should be left toward the end of the refining period after the bulk of the selenium has been removed with soda. Selenates are reduced in the presence of hydrochloric acid, but chloride ions are not desirable in lead-lined equipment and require rubber or plastic lining.

Not all the tellurium is precipitated on neutralizing, and eventually it appears in the refined selenium as a persistent impurity.

Notwithstanding the above, this process is simple and economical and is employed at several refineries.

In a patented Russian method[42] slimes, after removing copper, are roasted in a muffle furnace at

500-900°C to drive off selenium and then smelted with 30 pct soda + 10 pct pyrolusite. It is not clear from the patent specification what advantage pyrolusite offers over niter.

Japanese methods[37, 38] employ smelting the slimes with metallic lead and alkalies, the noble metals combining with the former and selenium with the latter.

Soda Roast: A modification of the pyrometallurgical method is the roasting or baking of raw or decopperized slimes intimately mixed with fine soda. The temperature must be kept below the sintering point to assure access of air essential to good elimination of selenium. The charge may be rabbled during heating. Selenium is recovered from the calcine water leach liquor by either of two methods.[39]

In the first method, it is neutralized to precipitate tellurium and then treated with sulphur dioxide, preferably in the presence of hydrochloric acid.

The second method was developed at Bolidens, where tellurium content is negligible. The leach liquor is evaporated to dryness and the sodium selenite and selenate reduced with coke to selenide.[40, 41]

$$Na_2SeO_4 + 3C \rightarrow Na_2Se + 3CO \qquad [10]$$

This is redissolved and blown with air to precipitate selenium and regenerate hydroxide, which is carbonated and returned to the process.

$$Na_2Se + H_2O + \tfrac{1}{2}O_2 \rightarrow 2NaOH + Se \qquad [11]$$

With proper roasting, selenium recovery may be 95 pct and over.

Any tellurium present in the slimes is similarly affected, forming a water soluble sodium tellurite. For this reason, soda roasting is not feasible with high tellurium slimes where recovery of high purity selenium is desirable.

The water leached slimes may be leached with dilute sulphuric acid. Good removal of copper is thus achieved.

Sulphuric Acid Roast: On oxidizing roasting of slimes, some selenium is driven off as dioxide or in elemental state and is recovered in the scrubber system. Selenium elimination is poor, the dust losses are high, the scrubber mud is impure and is usually returned to the process.

Much improvement is obtained with a sulphating roast as described in the section under copper removal. The slimes are mixed with acid prior to roasting. In C.C.R. practice, the mix is digested before charging into the roaster. Selenium dioxide is collected in the wet scrubber-Cottrell system, the combustion gases being led to a separate scrubber system. Some selenium is precipitated in the scrubber by the sulphur dioxide gas from the decomposed sulphuric acid. Both the selenium sludge and the scrubber solution are quite pure and yield a good grade of refined selenium.[20, 36]

A variation of this process was practiced at Outokumpu until recently, where the amount of selenium is too small to warrant construction of a Cottrell. The roaster gases were passed through a caustic soda solution to form sodium selenite and sulphite. Selenium was recovered on acidifying the solution with sulphuric acid.

Miscellaneous Methods: One of the earliest sources of selenium was the flue dust from sulphuric acid chambers. The flue dust is dissolved in hydrochloric acid and sodium and potassium chlorate. The acidity of the solution is brought up to 50 pct HCl and selenium recovered by reduction with sulphur dioxide. High acidity is necessary to keep tellurium in solution. The solution after recovering selenium is diluted and tellurium precipitated by sulphur dioxide.

This process is still in use at the Norddeutsche Affinerie, where the raw materials are Cottrell mud and flue dust. It is also the most commonly used method of selenium analysis.

A variation is the direct reduction of the flue dust with carbonaceous matter or[42] passing gases from roasted flue dust or slimes through hot carbon.

Leaching or boiling slimes with caustic soda is also practiced.[12, 43, 44] A complex liquor containing selenides, selenites, tellurides and tellurites is obtained. Treatment of such liquor is somewhat messy.

Another wet reagent is aqueous sodium sulphite. It dissolves free selenium only, forming sodium selenosulphate Na_2SeSO_3. The selenium is recovered by acidifying the solution. Only decopperized slimes contain some free selenium. The remainder of the selenium is combined and is not amenable to sodium sulphite leaching.

One patented method[45] employs distillation with hydrobromic and sulphuric acids. It is not known whether this has ever been attempted on commercial scale.

Passing oxygen through heated slimes as a means of forming volatile selenium dioxide has been attempted, but the results were unsatisfactory. Treating the heated slimes with gaseous chlorine was referred to in an earlier section.

Removal of Tellurium:[31] Tellurium is eliminated from the slimes as a water soluble sodium tellurite. It is converted into this form after a preliminary oxidizing or sulphatizing treatment. That may be done in several ways: (1) roasting or baking a slimes-soda mix—this is a combined oxidizing and "alkalizing" treatment; (2) refining with soda in Doré or cupelling furnace; (3) boiling the slimes with caustic soda after an oxidizing or a sulphating roast.

Sodium tellurite is extracted by water leaching the roasted product of (1) or the soda slag from (2). In all three cases the liquor also contains sodium selenite. Boiling the sulphate roasted slimes extracts some lead in addition to selenium and tellurium.

Whatever the method, the aqueous solution is neutralized to pH 6-6.2 to precipitate flocculent tellurium dioxide, or tellurium mud.[46] This is purified by redissolving in caustic soda, precipitating the impurities with sodium sulphide or sodium sulphite and neutralizing again to obtain tellurium dioxide of higher purity. In a method formerly used at C.C.R., the mud containing copper and selenium as impurities was roasted with sulphuric acid to volatilize the selenium and then water leached to remove the copper.[46]

There are three methods for recovering elemental tellurium from the mud: (1) direct reduction by heating with flour under a borax cover; excessive fuming of tellurium dioxide is a disadvantage; (2) dissolving the mud in hydrochloric acid or in sulphuric acid to which some salt is added, followed by reduction with sulphur dioxide, filtration, washing, drying and melting"; (3) dissolving the mud in caustic soda and electrolyzing with iron anodes and stainless steel cathodes.[47, 48]

The last method appears to be the most attractive of the three.

Parting and Recovery of Silver and Gold: As noted earlier, the purpose of a slimes treating pro-

cess is the removal and, where possible, the recovery of the impurities and the production of Doré metal.

At one time Doré metal was cast in thin sheets and boiled with sulphuric acid. The silver sulphate solution was diluted and silver recovered by precipitation on copper. The silver bullion assayed about 998-999 fine and always carried some gold and copper.

The electrolytic methods which displaced acid parting have been thoroughly described in the literature. There are two such methods—the Moebius, employing vertical electrodes and the Thum-Balbach, with horizontal electrodes. Examples of both are given in the second part of this paper.

Silver is deposited in the form of loosely-adhering crystals which are washed and melted into bars assaying 999.8-999.9 fine. The anode mud may contain 1/3 silver and 2/3 gold. It is boiled with sulphuric acid to dissolve silver and other impurities and to bring up the fineness. Lead and tellurium are not easily removed by boiling with acid and great care must be taken to eliminate them in furnace refining.

Should platinum group metals be present in sufficient quantity, the gold is refined electrolytically by the Wohlwill process. An example of this is also described below. Electrolytic gold is 999.8-999.9 fine.

The treatment of the platinum group metals is outside the scope of this paper.

To close the remarks on parting, "This step is in a very satisfactory state of development and but little improvement in methods is to be looked for" (Ref. 23 p. 123).

Silver Refinery Practice

Herewith are presented the data supplied by cooperating refineries. Analyses of anode copper, raw slimes and treated slimes are given in tables I, II and III, respectively. Flowsheets are shown for each refinery.

Bolidens Gruvatiebolag, Rönnskar, Sweden: Boliden Mining Co. produces anode slimes of high selenium-low tellurium content, (21 pct Se, 1 pct Te). Because of this combination, the first step in the flowsheet is a soda ash roast which is followed by a water leach to eliminate the selenium (fig. 1).

0.45 ton Na_2CO_3 is mixed per ton dried raw slimes. The mixture is roasted in a small multiple hearth furnace, at a maximum temperature of 840°F. Since temperature control is important, the furnace is electrically heated by resistors fixed to the hearths.

The soda-baked calcine is water leached in propeller-agitated tanks to remove selenium. The leach solution as press-filtered contains about 52 gpl Se and 32 gpl Na_2CO_3. This solution is treated for selenium recovery.

The copper content of the slimes is oxidized during the soda roast and is removed by leaching with weak sulphuric acid, following the water leach. Normally 0.78 ton concentrated sulphuric acid is required per ton dry raw slimes. The acid extracts most of the copper. The solution containing about 62 gpl Cu, 3.6 gpl Ag, 32 gpl H_2SO_4 is sent to insoluble anode tanks for copper recovery.

The treated slimes contain only 2 pct copper and 2 pct selenium and are smelted in an oil-fired Doré furnace in the usual way with a flux of 140 lb soda ash and 80 lb borax per ton slimes charged.

The Doré metal containing 858 silver, 106 gold, 25 Cu parts per 1000, is cast into anodes for parting in Moebius cells.

The Bolidens method of selenium treatment is unique and is claimed to produce a high purity selenium. The water leach of the soda roasted slimes is evaporated on a drum to give sodium selenite, selenate and carbonate. The mixture is reduced with coke in an electric resistance furnace, producing sodium selenide. This is dissolved in water and oxidized with air. In this reaction about 90 pct of the selenium content of the solution is precipitated in elemental form and NaOH is produced. The elemental selenium is removed and the solution containing the remaining 10 pct of selenium is carbonated to reduce the pH, and is then aerated to precipitate additional selenium. The solution is cooled to crystalize the soda ash, which is re-used in the roast of the raw slimes or for other purposes.

Canadian Copper Refiners Limited, Montreal East, Quebec, Canada: The C.C.R. process is described at greater length and with greater detail merely because of a more intimate acquaintance with it (fig. 2).

C.C.R. slimes are characterized by high selenium and tellurium content (20 pct and 3.5 pct). The present processing of tank house slimes can be divided into eight steps:

1. Drying, digesting and roasting slimes with sulphuric acid for selenium elimination, and conversion of copper to water-soluble sulphate.
2. Water leaching of roasted slimes to remove soluble copper, and cementation of silver from leach liquor.
3. Caustic-soda leaching of water-leached slimes to remove tellurium.
4. Refining of caustic-leached slimes to Doré bullion.
5. Parting of gold and silver by Moebius system.
6. Production and refining of selenium.
7. Recovery of copper sulphate from slimes water-leach liquors and discarded tank house electrolyte.
8. Recovery of acid from spent solutions from copper sulphate plant.

Raw slimes are pumped from the tank house to the silver refinery. They are delivered at a uniform rate to a thickener, from which they discharge to an Oliver continuous filter. The thickener overflow and Oliver filtrate are returned to the tank house.

The filtered slimes discharge directly to the top hearth of an 8 ft diam six-hearth oil-fired Herreshoff furnace. The furnace dries the slimes at 250° to 300°F to 5 pct moisture content, which helps to prevent dusting during subsequent handling.

The slimes-digesting equipment and chain-roasting furnace are arranged in such a way that after the dried slimes are fed to the digesters, gravity flow of material is possible to the point where the roasted slimes are discharged from the chain furnace.

There are two sets of three digesters each, arranged in line on either side of a central working platform. The digesters are 5 ft 0 in. diam by 15 in. deep inside. Each digester is provided with a set of four motor-driven rabble arms for agitating the slimes-acid mix, and a slimes outlet set tangentially with the outside of the shell to aid in emptying. The digesters are set in an insulated brickwork furnace to utilize the waste heat in the combustion gases from the chain roasting furnace.

Digestion of slimes is a batch process, 500 lb of dried slimes containing 5 pct H_2O being charged at the start of the process. Sufficient acid is added at the same time to produce a mix of good fluidity. The

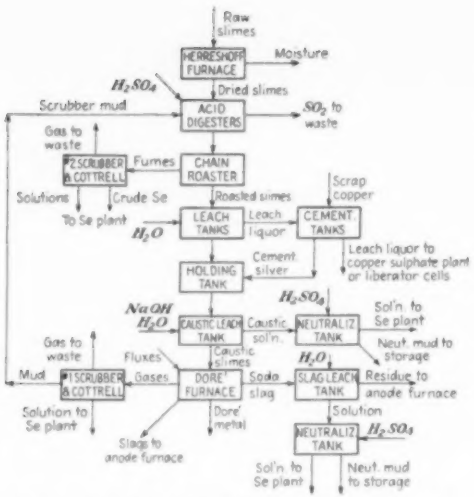

Fig. 2—Canadian Copper Refiners. Silver refinery flowsheet.

temperature of the mix is approximately 300° to 450°F, depending on the stage of digestion.

For typical raw slimes containing 32.90 pct insoluble Cu, 16.70 pct Ag, 20.83 pct Se and 3.76 pct Te, 1.74 lb of 100 pct sulphuric acid are required per lb of dry slimes. This is equivalent to 1.87 lb 66° Bé. acid. In practice, 2.23 lb are used, or 19 pct in excess of theoretical requirements.

At the conclusion of the digestion, the slimes mix is discharged to the feeder bin. From there it is fed by screw conveyor directly onto the pans of the chain roaster. The slimes bed thickness can be controlled and is normally ¾ in. thick at the start of roasting.

Essentially, the chain roaster installation consists of a furnace built around a pan conveyor on which the slimes mix is roasted to volatilize selenium content. The furnace brickwork measures 28 ft 2 in. x 4 ft 1½ in. x 5 ft 3 in. high, inside dimensions. The effective roasting area of the slimes bed in the roaster is 92 sq ft. Heat for roasting is supplied by a series of burners utilizing vaporized fuel-oil. The temperature of roasting is maintained at 800° to 900°F, which is sufficient to volatilize SeO_2 from the digested slimes, but not TeO_2. The time of roasting can be controlled between 30 and 60 min. With a slimes bed of ¾ in. and a roasting period of 45 min, the chain roaster production is 12,000 lb of well roasted product per 24 hr.

Near the discharge end of the furnace, a spiral cutter loosens the roasted cake from the pans, and the roasted slimes discharge into the calcine bin from which they are drawn into dump-bottom hoppers.

The roaster fume is drawn to the four-tower selenium scrubber system. Scrubber solution is pumped through sprays in the towers to humidify the gas prior to Cottrell precipitation and to recover the greater part of its selenium, selenium dioxide and sulphuric acid content. The products from the scrubber system are crude selenium and scrubber solution.

The Cottrell treater consists of two pipe units in parallel, treating 1800 cfm at 95°F.

The roasted slimes discharged from the chain roaster have the following assay: Total Cu 21.9 pct, water insoluble Cu 1 pct, Se 1.6 pct, Te 2.2 pct, Ag 2800 oz per ton, Au 260 oz per ton. Selenium elimination during roasting varies from 87-93 pct, depending on the selenium content and quantity of material fed. After completion of the new roaster, better performance is expected.

To dissolve the water-soluble copper sulphate the roasted slimes are leached in lead-lined agitator tanks. Countercurrent water leaching and decantation are employed with live steam being used for heating. Silver in solution is precipitated in a separate tank by cementation on copper anode scrap. The cemented silver is combined with the water-leached slimes in the holding tank.

Leach liquors are transferred to the copper sulphate plant or the liberator system. About 20 pct of the tellurium in the roasted slimes remains dissolved in the leach liquor after cementation on scrap copper.

The water-leached slimes are discharged to a holding tank from which they are pumped to a vacuum filter. The filtrate is returned to the water-leach liquor storage tank.

The leached slimes are discharged from the filter directly into another leaching tank which contains 10 pct caustic soda solution. An additional 50 pct of the tellurium originally present in the roasted slimes is removed by the caustic treatment, increasing the overall extraction to 70 pct. Simultaneously, 40 pct of the lead content is similarly removed, probably as sodium plumbite.

At the conclusion of the leach, the slimes and the liquor are pumped through a filter press. The filtrate is neutralized with sulphuric acid to remove tellurium and lead from the plant cycle as a tellurium mud which is presently being stored, pending retreatment at a later date for production of elemental tellurium. The caustic-leached slimes are retained in the filter press until they have been well washed with hot water and blown with compressed air to aid moisture removal.

Caustic-leached slimes are handled in batches containing 30 pct H_2O. A typical assay is Cu 3.7 pct, Ag 11,900 oz per ton, Au 1200 oz per ton, Se 6.5 pct, Te 3.3 pct. These slimes are charged to the Doré furnace for refining to metal for Moebius-cell parting.

The Doré furnace is of the reverberatory type and is fired with bunker C fuel oil. The hearth area is 34 sq ft and the uptake area 5½ sq ft. The hearth, side-walls and the facing on the bridge wall are built of magnesite brick. The rest of the furnace, including the roof and the verb, is constructed of fireclay brick.

The furnace is charged through a door in the sidewall and is skimmed from the front. About 12,000 to 15,000 lb of slimes containing 30 pct moisture, is mixed with fine soda and silica. This is charged to the furnace 500 lb at a time. When the charge is completely molten, the furnace is opened and No. 1, or siliceous slag, is skimmed off. This slag, which contains most of the arsenic, antimony and iron, is crushed and returned to the copper-anode furnace.

The charge is blown with air, lime being used as a flux. A slag of high lead content is next skimmed off and also returned to the anode furnace.

Fused soda ash is then fed to the furnace, and the charge is blown with compressed air. The soda slag formed contains most of the selenium and tellurium from the charge. This slag is leached with water to

dissolve the selenium and tellurium content. The residue from the slag leach is returned to the anode furnace. The alkaline slag leach filtrate is neutralized with sulphuric acid which precipitates tellurium, while the selenium remains in solution and is recovered at the selenium plant by SO_2 precipitation. The neutralized mud is also being stored at the present time.

The final stage of Doré furnace refining is the removal of the residual copper content by rabbling the charge with niter. The copper slag is skimmed off, crushed and returned to the anode furnace.

Just prior to casting, the Doré metal is cleaned with Portland cement. The slag from this is recharged to the furnace with the subsequent charge. Very careful refining of Doré metal is essential to meet the rigid selenium specification on premium silver bullion. Since silver bullion must contain not more than 3 parts per million selenium, Doré metal is not acceptable for parting if more than 500 parts per million selenium are present. Should the Doré contain more than that, it is re-charged to the furnace for additional refining.

The Doré metal is ladled into water-cooled anode molds by hand, and is treated for separation of silver and gold in the parting plant.

The gases leave the Doré furnace, pass through a flue for cooling and removal of much of the flue dust, and are then exhausted to the scrubber system. Besides the Doré furnace gases, all gases from the Hereshoff furnace and parting plant furnaces, and combustion gas from the chain furnace, are exhausted through the scrubber system for conditioning prior to Cottrell precipitation.

In passing through the three-tower scrubber system about 90 pct of the remaining dust is precipitated as a scrubber mud. Scrubber mud and solution are filtered through a lead-plate filter press. The mud is treated with raw slimes in the digesters. The scrubber solution is pumped to the selenium plant for recovery of selenium and sulphuric acid.

The gases leave the scrubber and pass to a three-unit Cottrell precipitator of the pipe type. The average metal loss per Doré charge from the Cottrell stacks is 6.7 oz Ag, 0.09 oz Au, 18.6 lb Se and 0.9 lb Te, as determined by continuous 24-hr sampling tests.

The selenium department is in a self-contained building.

Four materials are received at the selenium plant for processing: crude selenium from the chain roaster scrubber, neutralized slag leach solution, and two types of scrubber solution. Typical assays of the materials are given below:

Material	Selenium	Tellurium	H_2SO_4
Doré scrubber solution	23.97 gpl	2.15 gpl	207 gpl
Chain roaster scrubber solution	17.62 gpl	0.03 gpl	249 gpl
Neutralized slag leach solution	93.70 gpl	0.47 gpl	
Crude selenium	99.12 pct	0.010 pct	

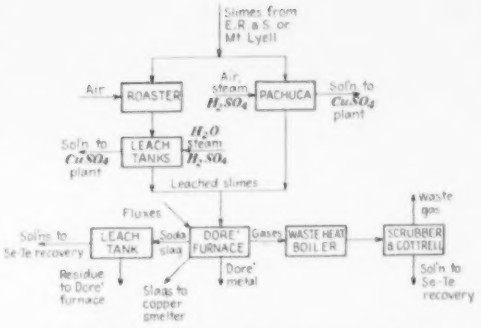

Fig. 3—E.R.&S. of Australia. Silver refinery flowsheet.

Silver refinery solutions are pumped to storage tanks at the selenium plant, from which they are drawn to precipitators as required. The precipitators, of which there are six in series, are lead-lined dished-end tanks, 5 ft diam by 8 ft 6 in. high. Each precipitator has a sulphur dioxide inlet and outlet, an inspection manhole, a solution inlet and a bottom discharge for solutions after precipitation is complete.

The six precipitators are connected in series in a closed circuit. Series precipitation under pressure aids in securing effective recovery of selenium with the minimum amount of sulphur dioxide. At the refinery, for every pound of sulphur burned an equal amount of selenium is precipitated, which indicates sulphur utilization of 80 pct. When a sample shows completion of precipitation, the precipitator batch is dumped to a settling tank. The solutions are boiled to expel sulphur dioxide. The selenium is allowed to settle, and the supernatant solution is decanted and pumped to the acid recovery system.

The crude selenium from the settling tank is ground, washed in a pebble mill and filtered in a vacuum wash box.

Sulphur dioxide is generated by burning elemental sulphur in a water-jacketed brick-lined steel tank. The gas discharges from the sulphur burner into a steel tank to settle elemental sulphur sublimed from the burner and to cool gases before they pass to a lead-lined water scrubber tank. Sulphur dioxide passes from the scrubber to the first precipitator in series on the floor above.

Crude selenium assays from 0.01 to 0.10 pct Te, depending on the source of the material. After washing and drying, the selenium is charged to the retorts in batches of 1000 lb. Retorting crude selenium serves three purposes: it drives out moisture, acid and other foreign materials; reduces the tellurium, iron and copper content; and recovers in the non-volatile residue any silver and gold present.

After every second charge, the retort is cooled,

Table IV. Typical Analyses of C.C.R. Slimes and Doré Materials

In Parts per 1000

	Raw Slimes	Roasted Slimes	Treated Slimes	No. 1 Slag	Lead Slag	Soda Slag	Copper Slag	Flue Dust	Scrubber Mud	Doré Metal	Silver Bullion	Gold Bullion
Cu pct	32.90	21.93	3.73	4.00	5.98	2.19	17.38	3.25	0.05	5		
Ag o/t	4870	2803	11,900	338	417	158	1798	1343	3212	905	999.9	
Au o/t	432	259	1200	6	4	7	30	78	84	85	0.002	997
Se pct	20.83	1.60	6.56	0.45	0.37	22.71	5.58	31.11	40.62		0.002	
Te pct	3.76	2.21	3.36	0.93	0.52	12.27	0.91	2.79	1.62			
Pb pct					21.70							

Table V. Typical Analyses of Doré Furnace Charge and Products

	Leached Slimes	Doré Fce. Slag	Soda-Niter Slag	Selenium Matte and Skimmings	Cottrell Fume	Doré Bullion
Cu	2.18	4.82	6.0	16.0		
Ag	3367	403	808	4571	1805	777.68
Au	897	13	16	1009	8	193.06
Se	0.70		2.0	0.26	3.9	
Te	4.35		6.8	2.38	2.9	
Pb	28.79	17.18		1.0		
Ni	0.39			0.42		
Fe				3.6		
S	5.62					
As	1.04	0.66		0.18		
Sb	9.60	12.31		2.4		
Bi	0.05	0.24				
CaO						
SiO_2				37.6		

removed from the furnace, and cleaned of any accumulated residue. The residue containing about 65 pct Se as iron selenide is ground, screened, digested with sulphuric acid and roasted in the chain roaster at the silver refinery.

The retort bowls are heated in an oil-burning furnace at a temperature just sufficient to distill the selenium (1200°F). The retort distillate is collected in blocks which are allowed to cool slowly to obtain the grey metallic variety.

Metallic selenium is broken in a small jaw crusher and then pulverized in a pebble mill which is operated in closed circuit with an inclined vibrating screen. All +200 mesh is reground in the ball-mill.

The Electrolytic Refining and Smelting Co. of Australia Proprietary Limited, Port Kembla, N. S. W.: E.R.&S. treats the slimes from the copper refinery at the Mount Lyell Mining & Railway Co. Ltd., Queenstown, Tasmania, as well as the slimes from its own anodes.

Mt. Lyell slimes contain much copper sulphide which necessitates roasting to the oxide, since copper sulphide is insoluble in sulphuric acid. Mt. Lyell slimes, if treated alone, tend to frit when roasted, due to high copper content—67 pct. It was found necessary to roast and leach twice to reduce the copper to a sufficiently low level for satisfactory Doré furnace operation. On the other hand, E.R.&S. slimes do not settle readily when leached alone. For these reasons, it is preferred to mix the slimes before treatment (fig. 3).

A coal-fired reverberatory furnace, with two superimposed hearths, 19 ft 6 in. x 9 ft 0 in., is used for roasting. The combustion gas passes over the top roasting hearth and under the bottom drying hearth.

Three different types of roasts are used, depending on the amounts of slimes to be treated and the labor available, as follows:

1. 14 hr roast. One-half of the furnace is discharged and recharged alternately every 7 hr. This is used for Mt. Lyell slimes, or a mixture of the two slimes.
2. 16 hr roast. The charge is in furnace for 24 hr, but rabbled during 16 hr only. This method is not used alone for Mt. Lyell slimes because of tendency to frit if not rabbled continually.
3. 24 hr roast. Higher proportions of Mt. Lyell slimes can be treated with this roast.

At completion of the roast, the slimes charge is quenched with water to prevent fuming and dusting and is raked from the roaster into portable hoppers for charging into the leaching tanks. These tanks are constructed of sheet lead on a skeleton steel framework and are equipped with two-bladed antimonial lead propellers. The necessary solution is first charged to the tanks, and then sulphuric acid is added. Live steam is used for heating and agitation is continued for 2 hr, at which time the copper should all be dissolved. Sodium chloride is used to precipitate any soluble silver. The solution is allowed to settle and is decanted through a charcoal-filled filter box to the copper sulphate plant for recovery.

The leached slimes are then filtered and dried on the lower hearth of the roaster, bagged and stored until sufficient are accumulated for a Doré furnace campaign. The leached slimes contain about 2 pct Cu, 11 pct Ag, 3 pct Au, 0.7 pct Se, 4 pct Te, 28 pct Pb.

When the quantity of slimes is in excess of the roaster capacity, a Pachuca is used to leach the excess. E.R.&S. prefers to use the roaster for all slimes, since some selenium, tellurium and arsenic are eliminated in this way. Also, because copper sulphide is unaffected by sulphuric acid, Pachuca leached slimes causes excessive quantities of matte in the Doré furnace.

E.R.&S. is presently engaged in improving its slimes treatment processes, and the above difficulty will be overcome.

The Pachuca consists of a lead-lined steel tank 3 ft 6 in. diam by 16 ft 0 in. high, fitted with a cone bottom. Live steam is used for heating. Compressed air used for oxidation provides all the agitation necessary to suspend the slimes. A charge of 1000-1800 lb slimes can be leached every 24 hr. The Pachuca slimes are smelted with regular leached slimes.

The Doré furnace is an oil-fired reverberatory. The removable hearth, 4 ft 1½ in. x 4 ft 0 in. is built of magnesite brick set on end with Grefco cement in an iron frame, which is held on lifting jacks and set on a track for ready removal. A paste of magnesite plus 5 pct silicate of soda is used for making a tight joint when a hearth is to be raised into place. When the hearth is in place it is fettled with 4 in. of magnesite. The inner furnace walls are magnesite and the arched roof is high-alumina.

The exit gases from the Doré furnace pass under a waste heat boiler which serves to cool the gases. Gases from all furnaces are passed to a water scrubber for humidification and then to the Cottrell.

When its specific gravity exceeds 1.05, the scrubber and Cottrell solution is withdrawn for selenium recovery. The residue collected in the scrubber is filtered, bagged and returned to the roasting furnace. Cottrell tube cleanings are treated in the same way.

Leached slimes are charged to the Doré furnace with soda ash, silica and niter (if Pachuca slimes are present) in approximately these proportions:

	Lb
Leached slimes	420
Selenium matte	50
Cottrell fume	50
Soda Ash	100
Niter	10-20 (if Pachuca slimes are being treated)

When the charge has melted, coarse charcoal is used to reduce metal from the slag which is then skimmed and another charge is added. Three charges are made per 8-hr shift until sufficient bullion has accumulated for refining. This takes 3 or 4 days. The

Doré slag is returned to the Matting Furnace Department.

The selenium matte which forms a layer above the bullion, is decomposed by rabbling with soda ash and niter. The soda-niter slag is treated for selenium and tellurium recovery by leaching, if it contains at least 12 pct tellurium. Otherwise, it is retreated in the Doré furnace or sent to the copper smelter, depending on its precious metal content.

Air is blown onto the surface of the bullion which is then cleared by thickening with ashes and skimming. The skimmings, known as "Selenium Matte" are recharged to the Doré furnace in the subsequent campaign. This process is repeated and finally the bullion is refined with niter until the fineness is about 980 and is then cast into anodes for parting in Balbach-Thum type cells of glazed porcelain. (Table V.)

Norddeutsche Affinerie, Hamburg, Germany: Norddeutsche's selection of slimes treatment flowsheet (fig. 4) is predicated on the proximity of the Norddeutsche Affinerie's own lead refinery. It is interesting to note that the Norddeutsche process is somewhat similar to that used by American Smelting and Refining Co., Barber, New Jersey, whose copper refinery is also close to a lead smelter.

There is considerable variation in the analyses of anodes being treated at Hamburg, as is shown in table I. This accounts for the varied analyses of slimes charged to the cupel furnaces.

Slimes from the electrolytic cells are settled and the supernatant electrolyte is returned to the tank house. The thickened slimes are pumped into a brick-lined oxidizer to which water and sulphuric acid are added to obtain 120 gpl H_2SO_4. The mixture is heated to 140°F with direct steam. Compressed air is blown through the tank to oxidize and dissolve the copper. Blowing is continued until the free acid content of the liquor becomes constant. The slimes are then settled and the clear liquor is returned to the tank house.

After leaching, the slimes contain only 2-3 pct copper, compared to 16-20 pct before treatment. The treated slimes are passed to a rotary vacuum filter and are dried on a drum dryer to 10 pct moisture. These slimes, which are the feed for the cupellation plant, contain:

Cu	2-3 pct	As	3-4 pct
Ag	408-612 oz per ton	Sb	5-7 pct
Au	26-131 oz per ton	Sn	1.1-4 pct
Se	4-10 pct	Fe	0.25 pct
Te	0.6-1.3 pct	SiO_2	5-9 pct
Pb	9-18 pct	Bi	0.4-1 pct
Ni	0.6-4.0 pct		

The slimes are processed with foreign slimes and lead containing 4½ pct silver plus gold, from the lead refinery. About 880 lb of lead is required per ton of dry treated slimes. The charge is oxidized by blowing with air, and produces 660 lb of rich lead and 2000 lb of slag per ton treated slimes. The rich lead produced in the first cupel contains 30-40 pct silver plus gold, 50 pct lead and 6 pct copper.

The cupel is made of rammed marl and lasts for about 25 charges. 33 tons of slimes are cupelled per charge. The cupellation takes 9 hr per ton of charge and consumes 1800 lb of coal per ton of slimes charged. The gas from the first cupellation is passed to a wet Cottrell for recovery of the metal content as a selenium mud which is later treated for selenium recovery.

The products of the second cupel are silver bullion

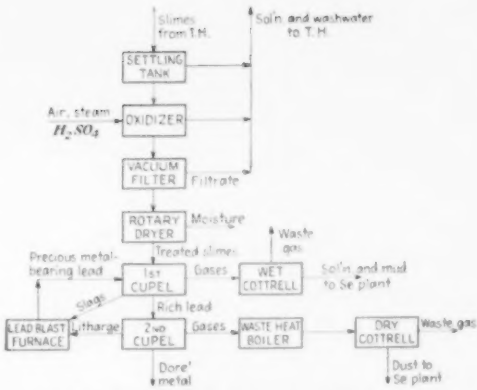

Fig. 4—Norddeutsche Affinerie. Silver refinery flowsheet.

containing 99 pct silver plus gold, and litharge containing 62 pct lead, 13 pct copper and 4 pct silver plus gold. The second cupel furnace can treat up to 10 tons of silver bullion per charge. The second cupellation requires 25 furnace hr per ton bullion. The cupel lasts for 15 charges and requires 2.8 tons coal per ton silver bullion produced.

The gas from the second cupellation is treated in a dry Cottrell. As a rule, the Cottrell dust is smelted in the lead blast furnace; when rich in selenium it is treated together with the mud from the wet Cottrell for selenium recovery, as explained later.

The bullion from the second cupel is cast into anodes measuring 10 x 8 x ¾ in. and weighing 320-350 oz. These anodes are electrolyzed in earthenware Moebius cells.

Selenium is recovered at Norddeutsche from three materials: the mud and solution from the wet Cottrell and the dust from the dry Cottrell. The analyses of these materials are:

Selenium Materials

	Se	Te	Pb	SiO_2	H_2SO_4
Cottrell solution, gpl	96				270
Cottrell mud, pct	25-31	1-5	16-18	3-4	
Flue dust, pct	20-23				

The Cottrell mud and flue dust are mixed and leached in a brick-lined tank with HCl (30 pct), H_2SO_4 and $NaClO_3$. Depending on the selenium content of the materials, they are leached two or three times, countercurrently. The leached residue is returned to the blast furnace for lead recovery.

Hydrochloric acid is added to the clear solution from the first leach and the selenium is precipitated with sulphur dioxide. The precipitated selenium is filtered, washed, dried, melted and pulverized. Should the tellurium content of the selenium precipitate be excessive, the selenium is redissolved in hydrochloric acid and sodium chlorate and reprecipitated with sulphur dioxide.

After removal of the selenium by filtration, the spent solution is diluted with water, and treated again with sulphur dioxide to precipitate tellurium. The tellurium is filtered, washed, dried, melted and pulverized. The solution after tellurium removal is neutralized with lime and discarded.

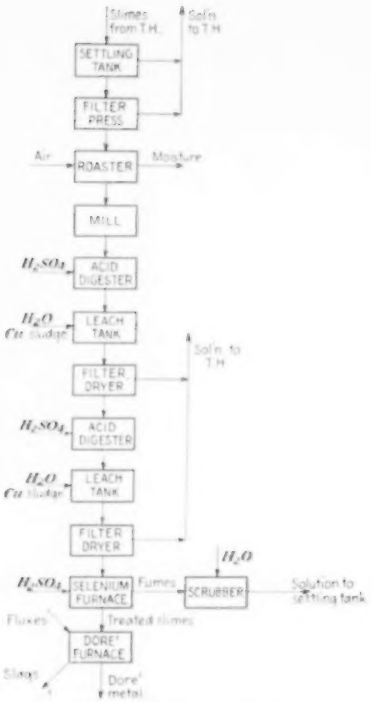

Fig. 5—Outokumpu Oy. Silver refinery flowsheet.

The overall chemical consumption per pound of selenium produced amounts to:

$NaClO_3$	1 lb
HCl, 19° Be	2-2.5 lb
H_2SO_4, 60° Be	1.2-1.5 lb
SO_2	2-2.2 lb

Outokumpu Oy, Pori, Finland: The anodes refined at Outokumpu have a very high nickel content (0.7 pct), so that the resulting raw slimes require an intensive treatment to remove nickel and copper prior to Doré furnace smelting (fig. 5).

The processing covers the following steps:

1. Drying and air roasting raw slimes to oxidize the copper content.
2. Sulphuric acid-digesting and water-leaching of roasted slimes to dissolve copper and nickel as sulphates.
3. Sulphatizing roast of leached slimes under reduced pressure to eliminate selenium.
4. Smelting of treated slimes to Doré metal.
5. Parting of gold and silver; recovery of platinum metals.

Raw slimes from the tank house are thickened by settling, filtered in a press and washed free of electrolyte. The raw slimes analyze: 11 pct Cu, 45 pct Ni, 9 pct Ag, 0.5 pct Au, 4 pct Se. The washed raw slimes are air-roasted in a stainless steel electrically-heated rotating drum, 2 ft 8 in. diam by 9 ft 10 in. long. Electrically heated air is circulated through the drum and cyclone by a fan. The drum rotates ¼ rpm and is maintained at a temperature of 660°F. The roasted slimes are milled to break up lumps, and are then digested with sulphuric acid.

The digesting is done in two electrically-heated cast iron troughs, set in brickwork and equipped with lead fume hoods. The temperature of the digestion is kept below that at which selenium dioxide vaporizes (603°F). The digestion is done in batches, slimes being added gradually to the bath of hot concentrated acid. The acid consumption is 1.2 lb per lb slimes.

Because operations are on a small scale, and all heating is done electrically, the gas volumes are very small, and the Outokumpu silver refinery is therefore not equipped with scrubber or Cottrell.

In the acid digesting of roasted slimes, the nickel oxide and copper are converted to water soluble sulphates. Some silver sulphate and selenium dioxide are also formed.

The digested slimes are leached in agitated tanks with hot condensate from the tank house. The leaching dissolves most of the nickel and copper plus part of the silver and selenium. The latter two elements are precipitated from the solution by cementation, on copper sludge, which is added to the leach batch. The liquor after cementation is filtered through a lead plate and frame press and returned to the tank house. The slimes as obtained from the press are called No. 1 leached slimes and contain about 6 pct Cu, 12 pct Ni, 27 pct Ag and 11 pct Se.

These slimes are dried, digested and leached a second time to remove almost all the copper and nickel. The liquor is treated as for the first leach above. The No. 2 leached slimes contain 0.5 pct Cu, 0.9 pct Ni, 36 pct Ag and 13 pct Se.

It is important that the nickel content of No. 2 leached slimes be kept low, as it has been shown that high nickel content of slimes results in low yield of Doré metal due to sticky slags.

The No. 2 leached slimes are filtered and charged into cast iron pans with a measured amount of concentrated sulphuric acid. The pans are then placed in an electrically-heated stainless steel muffle furnace. The selenium dioxide and sulphur dioxide are drawn from the furnace by suction and are absorbed in weak sulphuric acid solution forming selenium and additional sulphuric acid.

The roasting operation is a batch process and produces refined slimes containing less than 1 pct selenium.

The solution from the absorption towers is returned to the settling tanks. Selenium is precipitated by sulphur dioxide and purified by distillation.

At Outokumpu the production of selenium is relatively unimportant but sulphatizing-roasting is carried out to facilitate Doré smelting.

The refined slimes with suitable fluxes are charged to a small oil-fired furnace. Smelting operations are simple due to the prior removal of copper, nickel and selenium, and slag losses are therefore kept low. The Doré slag is charged to the copper anode furnace.

Doré anodes contain:

	Pct		Pct
Cu	1.12	Pt	0.02
Ag	92.69	Pd	0.05
Au	4.92	Bi	0.01
Se	0.03	Pb	0.006

These anodes are electrolyzed in Moebius cells.

Rhodesia Copper Refineries Limited, Nkana, Northern Rhodesia: Rhodesia Copper Refineries Ltd. is owned jointly by Rhokana Corporation Ltd. and

Table VI. Comparative Operating Data

Refinery		C.C.R.	E.R.&S.
Type of Cell		Moebius	Balbach-Thum
Electrolyte			
	Ag, gpl	100	90-120
	Cu, gpl	15	60 max.
	Free HNO_3, gpl	2	3-10
Circulation		Stirring by mechanical scraper	None
Current			
	Current per cell, amp	500	200
	Current density, amp per sq ft	30 cathode 44 anode	40 anode
	Voltage per cell, volts	2.8	5.0
	Current efficiency, pct	93.5	?
	Kw hr per Troy lb Ag	0.276	?
Anodes			
	Composition, ppt	905 Ag, 85 Au, 5 Cu	778 Ag, 193 Au
	Size	9 in., 6 in., ½ in.	10¼ in., 7½ in., ¼ in.
	Weight, troy oz	150	70
	No. per cell	15	10
	Life	28-34 hr	2-3 days
	Method of support	By clip from bus bar	In a cradle
	Anode enclosure	Bag	Cradle, 2 per cell
	Anode enclosure	21¼ in., 2 in., 15¾ in.	26 in., 9¼ in., 7¼ in.
	Anode scrap, pct	25	?
Cathodes			
	Number per cell	6	4
	Material	Rolled silver sheets	Carbon strips
	Size	20 x 12 x 1/16 in.	24 x 5 x ½ in.
Cells			
	Material of construction	Steel, rubber-and-brick-lined	Glazed earthenware
	Number	15	20
	Size	30⅞ x 30⅞ x 24 9/16 in.	49 x 24 x 9¼ in.
	Cell arrangement	3 sets of 5 in series	20 in series
	Silver removed after ? hr	12	?

Nchanga Consolidated Copper Mines Ltd. and refines the entire copper output of these two companies.

Because only 2.8 lb of raw slimes are produced per ton of anodes refined, slimes are cleaned from electrolytic cells only once every 100 days and sent to another refinery for treatment. The refinery capacity at Nkana is being increased to 12,000 tons per month, and it is probable that a treatment plant will be ultimately built to recover Au, Ag, Se and Te. Judging by their analysis, Nkana raw slimes would be well suited to decopperization by aeration in weak sulphuric acid.

Parting Plant Practice: As noted in the introduction, the second portion of the paper describes methods of treatment of raw slimes up to the production of Doré metal. To complete the data, this final section covers electrolytic parting plant practice in general, although figures for two actual plants are used as illustrations.

Doré metal is parted electrolytically in either Balbach-Thum or the more generally used Moebius cells. Comparative operating data are given in table VI.

The Moebius cells are arranged in 3 groups of 5 cells each. Fifteen anodes are suspended in each cell, so that the total loading for 15 cells is 34,000 troy oz. Each set of three anodes is enclosed in a filter duck bag. On parting, the silver dissolves in the electrolyte at the anode and plates out as a nonadherent deposit on the cathode. The gold from the anode is collected as an insoluble mud inside the bag.

In each Moebius cell is suspended a wooden frame basket having a filter cloth bottom. Continually reciprocating wooden scrapers brush the silver crystals from the cathodes into the basket filter bottoms. The baskets are raised from the cells every 12 hr for removal of the silver crystals, which are washed with water in a filter-bottom stainless steel cart to remove all traces of electrolyte. The washed silver sand is dried and melted in a silicon-carbide retort, 15,000 oz at a time. Silver bullion is cast in 1000 oz bars, assaying 999.8+ fine Ag, 2 ppm Au, 2 ppm Se and less than 1 ppm Pd.

Anodes are removed from the Moebius cell bags after 24 to 28 hr, scraped to remove the passive gold film and replaced in cells for an additional 4 to 6 hr. When necessary, scrappy anodes are replaced in cells by new ones. Once every four days all anodes are removed, the bags are drained and emptied and the scraped anode scrap is remelted in a graphite crucible and cast into Doré anodes for parting.

The gold mud scraped from the anodes is combined with that from the bags, washed with water and boiled with concentrated sulphuric acid to remove silver and selenium. The acid leached gold is washed with water and drained on Filtros blocks. The gold sand is melted in a crucible furnace and cast into bars assaying 998+ fine Au.

In each Balbach-Thum cell, two acid-proof painted wooden cradles are immersed 1½ in. in the electrolyte. The cradle bottoms are covered with 12 oz duck and five anodes are placed in each cradle, contact to them being made with a silver stud. The anodes in the two cradles in each cell are in parallel, while the cells are in series circuit.

The cell bottom is covered with carbon strips which serve as the cathode. One end of the cell slopes from the bottom up to the front edge which facilitates silver crystal removal.

Silver crystals are removed from the cathode by raking them up the sloping bottom. After the electrolyte on the silver has drained back to the cell, the crystals are transferred to a mobile filter box, and washed as described above.

The gold mud is cleaned from the cradles daily after the first two days of each campaign, washed

Table VII. Details of Wohlwill Cells

Type of Cell		Wohlwill
Electrolyte		
	Au	70-90 gpl
	HCl	150
	Temperature	150°F
Circulation		By glass propeller
Current		
	D.C. per cell	90-95 amp
	Current density	50 amp per sq ft
	A.C. per cell	105 amp
	Voltage per cell	1.5 v
Anodes		
	Composition	998 Au, 1.5 Ag, 0.5 Pt, 1.0 Pd, parts per 1000
	Size	6 x 2¾ x ¼ in.
	Weight	35 oz each, 315 oz per cell
	No. per cell	9
	Per cent scrap	25 pct
Cathodes		
	Material	Pure gold
	Size	11 x 2¼ x 0.015 in.
	No. per cell	11
Cells		
	Material	Glazed earthenware
	No. of cells	4
	Volume per cell	20 liters
	Heating	Electrically heated water jackets

with water and leached with sulphuric acid. The leached gold contains sufficient platinum metal and is melted in a graphite crucible and cast into anodes for refining in Wohlwill cells.

Details of the cells used at E.R.&S. are given in table VII.

The anodes and cathodes are in parallel in each cell, and the cells are in series.

In the electrolysis, gold is dissolved from the impure anode and deposited on the pure gold cathode. A.C. is superimposed on the D.C. to flake the silver chloride off the anodes, which would otherwise tend to become passive. The silver chloride collects with some gold as a sludge at the bottom of the cell. This is used for electrolyte makeup, by dissolving in aqua regia, boiling off the nitric acid and filtering. The filtration removes silver chloride and other impurities, which are melted to Wohlwill residue bullion and retreated in the Doré furnace.

Palladium and platinum dissolve in the electrolyte. Periodically, a portion of the electrolyte is withdrawn for platinum metals recovery, and is replaced with fresh gold chloride solution.

As each cathode attains a weight of 60 oz, it is removed, washed in cold water and drained. The cathodes are melted and cast into bars which assay 999.7 Au, 0.2 Ag parts fineness. The anode scrap from the Wohlwill cells is remelted daily and cast into new anodes.

Scrap from both Balbach-Thum and Wohlwill cells is worked up in a diminishing number of cells until the particular batch or campaign is worked out almost completely. The final scrap is then insufficient to run one cell.

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Alkaline selenites and selenates evaporated to dryness and reduced with carbon to selenides, dissolved and blown with air to form selenium. Solution recycled.

[46] Oliver C. Martin and Charles W. Clark. U. S. 2,076,738, Apr. 13, 1937. Tellurium recovery. Water leach liquor of soda slag neutralized with sulphuric acid, the precipitated tellurium mud roasted with sulphuric acid to drive off selenium, water leached to recover copper, dissolved in 20 pct sulphuric acid and reduced with salt and sulphur dioxide to precipitate tellurium.

[47] A. J. Phillips, Y. E. Lebedeff and J. R. Crandall. U. S. 2,413,374, Dec. 31, 1946. Process for producing elemental selenium. Flue dust, slimes, etc. roasted in a retort at 1350°F. Selenium and selenium dioxide-containing gases pass through hot carbon to reduce the dioxide.

[48] F. F. Poland. U. S. 2,048,563, July 21, 1936. Recovery of selenium. Slimes decopperized then oxidized in presence of molten caustic alkali.

[49] Tsutomu Shiomi and Kenzo Ozu. Jap. 36,939, Aug. 11, 1920; *Chem. Abstracts* (1921) **15**, 3728. Selenium from copper slimes. Slimes leached with warm caustic soda, residue heated with potassium nitrate and leached with caustic soda or sulphuric or hydrochloric acid.

[50] Wesley G. Woll and Robert T. Gore. Production of Tellurium. U. S. 2,258,963, Oct. 14, 1941. Tellurium deposited electrolytically from an alkaline solution on stainless steel cathodes.

Side-blow Converter Process

for the Production of

Low Nitrogen Steel Ingots

by R. R. Webster and H. T. Clark

The side-blown converter has been investigated as a possible commercial process for the production of low nitrogen steel. During this work, two converters of 3-ton and 22-ton capacity were operated on a pilot plant basis for a total of 214 heats. The steel made in these converters was low in nitrogen and possessed good cold working properties. Some problems of converter operation remain to be solved.

IN plants operating with a high iron capacity, several different refining methods are used in the conversion of the molten pig iron to steel. These include various ore practices in stationary and tilting open-hearths, the duplex process employing the Bessemer converter and open-hearth, and the Bessemer process.

At J&L, a considerable part of the iron produced is handled by the Bessemer process, either alone or in conjunction with duplexing, and therefore an appreciable portion of the steelmaking research effort has centered about the method. This paper covers research work on the development of the side-blow converter for the commercial production of low nitrogen ingots and includes descriptions of the operation of a 3-ton and a 22-ton experimental converter at the Aliquippa Works.

The refining of iron to produce steel requires the removal of a large portion of the carbon and silicon and the control of manganese, phosphorus and sulphur which are present in the iron in varying amounts. The first large-scale means of refining iron was the acid Bessemer process which was brought into use almost 100 yr ago. This method, using compressed air as the refining medium, accomplishes substantially complete removal of carbon, manganese and silicon. Phosphorus and sulphur are not affected but, by choice of an iron composition sufficiently low in these elements, a commercial product can be produced. Since the process will handle large tonnages rapidly, operates without external fuel and with a minimum of additional equipment, it quickly became the major tool in the early expansion of the steel industry. Later, the basic open-hearth process, by affording control of phosphorus and sulphur and by consuming the large quantities of steel scrap that were becoming available, forced the acid Bessemer process into a secondary position in the industry. During the past two decades the demand for steel to be used in cold forming and drawing operations has gradually increased. Bessemer steel, because of its work hardening and aging characteristics, is not as suitable for these applications as basic open-hearth steel, consequently the decline of the process was accelerated. More recently, because of changing economic conditions, this long range trend appears to have been arrested or perhaps reversed. Ingot production data for recent years furnishes only an incomplete picture of the importance of the converter in the American steel industry; open-hearth furnaces utilize large tonnages of blown metal for which no published statistics are available.

Metallurgical Aspects

The fundamental difference between Bessemer and open-hearth steels apparently lies not in the method of manufacture but, rather, in the differences in chemical composition of the two steels. It is further believed that the principal features distinguishing Bessemer from open-hearth steel are the higher nitrogen and phosphorus contents of the former.

Evidence supporting this position is supplied by tests on laboratory induction furnace heats that were made to contain varying amounts of phosphorus and nitrogen but were otherwise similar to normal low carbon silicon-killed steels. Fig. 1, 2 and 3, summarizing the test results, are taken from G. H. Enzian's paper titled, "Some Effects of Phosphorus and Nitrogen on the Properties of Low Carbon Steels."[1] Fig. 1 indicates that phosphorus has a marked effect on the cold work embrittlement of steel as shown by the work brittleness test of Graham and Work.[2] In the low nitrogen steels, which as a group have the better cold working properties, the effect of phosphorus variations is the more pronounced.

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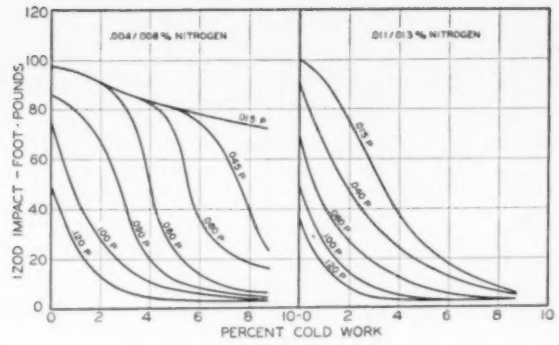

Fig. 1—The effect of phosphorous on the embrittlement of steel by cold work. (Enzian.)

Fig. 2 gives similar information using nitrogen as the major variable. Here, the response in the cold working properties of the steel to changes in nitrogen content is evident. It will be noted that the effects of phosphorus and nitrogen are additive and that in the case of low phosphorus steel there is a major change in properties at about 0.011 pct nitrogen.

Fig. 3 shows that aging, as indicated by increase in tensile strength at 500°F, is a function of nitrogen content. Hot tensile test[3] results demonstrate that in the lower range, any change in nitrogen is quite effective in altering the aging of steel. Above about 0.008 pct nitrogen, the steel has strong aging characteristics and the effect of additional nitrogen is not pronounced. Phosphorus does not appear to contribute to aging and both high and low phosphorus steels were used in preparing the figure.

The effect of oxygen and hydrogen on the properties of Bessemer steel have been considered but there is no substantial evidence that they are major contributing factors.

Preliminary Work

As early as 1939, study of the Bessemer process had shown the way to possible improvement of its product. The work was directed primarily toward nitrogen control[4, 5] since phosphorus is normally handled by duplexing and, should the problem arise, it may be removed in the converter shop by the Yocum[6] and Perrin[7] processes or by final treatment in a basic lined converter.

Nitrogen absorption in the bottom-blown converter was found to be dependent on (1) operating temperature, (2) length of blow, and (3) bath depth. Also, by increasing the oxygen concentration in the process, that is, the substitution of ore for the cold scrap used in the control of blowing temperature, nitrogen was lowered. The use of several of these methods simultaneously resulted in a very considerable reduction in nitrogen. However, the permissible changes in operating conditions are controlled largely by the design of the converter shop so that under satisfactory operating procedures the improvement in nitrogen content was not sufficient.

Parallel with the work on the bottom-blown converters, experiments with small foundry-type side-blown converters (fig. 4), demonstrated that these units could produce steel with a nitrogen content comparable to that of the scrap open-hearth process. Within the limits of the tests, the nitrogen content of steel from the side-blown converter showed no dependence on blowing temperature or blowing time.

No information was available on the physical properties of unkilled steel made by the side-blown converter so, on special order, a rimmed ingot was made at a steel foundry.

Table I. Composition and Physical Properties of Experimental Side-blown Steel

Chemical Composition in Percent				
C	Mn	P	S	N_2
0.11	0.31	0.045	0.031	0.004

Physical Properties of Normalized Steel	
Tensile Strength	58,170 psi
Yield Point	46,680 psi
Elongation	38 pct
Reduction in Area	67.7 pct
Hardness—Rockwell B	61
Brinell Hardness Number	115
Izod	95 ft lb

This ingot was rolled into bars and tested. The composition and physical properties of the heat are shown in table I while fig. 5 gives the work brittleness results. The properties are sufficiently good that it is comparable to duplex steel rather than Bessemer which, incidentally, would fall in the lower third of the shaded area.

Investigation of the literature[8, 9] disclosed that even before 1900, side-blown converters had been used for the production of ingots and that processors had claimed superior cold working properties for such steel. This position was not generally accepted and the side-blow process was relegated to the foundry where it is still used today, largely because it will produce very high temperature blown metal.

Design and Operation of the 3-ton Experimental Converter: These preliminary investigations with bottom-blown, mill converters, and side-blown foundry converters indicated that a considerable degree of control over the nitrogen content of Bessemer steel can be exercised by changes in steel-

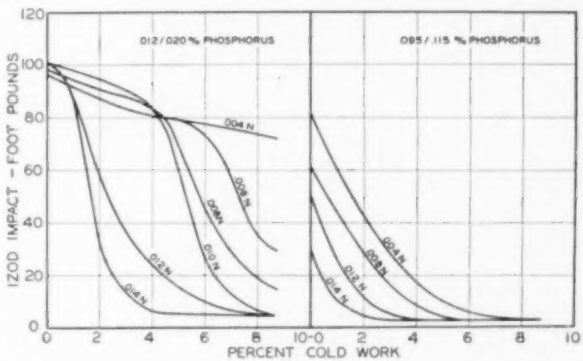

Fig. 2—The effect of nitrogen on the embrittlement of steel by cold work. (Enzian.)

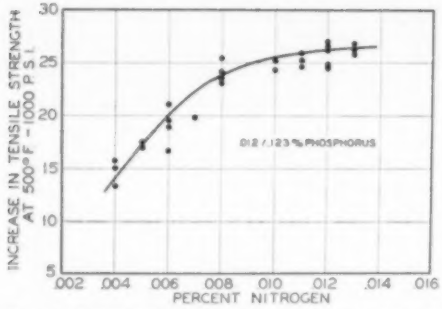

Fig. 3—The effect of nitrogen on the aging characteristics of steel.

(Engineer)

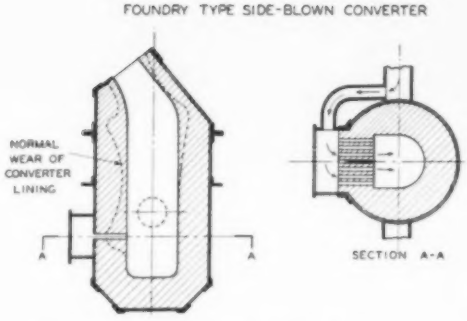

Fig. 4—Foundry type side-blown converter.

making practices. However, radical changes in either design or operation could not be made on the mill converters without undue loss in production and excessive costs. It was evident early in the work that more rapid progress would be made if an experimental unit were available so that variations in blowing practice, etc. could be made at will. The following considerations were believed to be of paramount importance:

1. The nitrogen content of the steel should be as low as possible.

2. The operation of the vessel should be efficient, with high yields of metal (little spitting or slopping), short blows, good refractory life, adequate blown-metal temperature, etc. The small foundry side-blown converter does not meet this requirement satisfactorily.

3. The unit should be capable of enlargement to full commercial size with a minimum of change.

Plans were drawn up, therefore, for a pilot converter which would make about three tons of steel. Into the design were incorporated features that would give maximum flexibility of operation. An acid lining was used; serious consideration was given to basic linings for phosphorus removal as in European practice, but this phase of the work was deferred until the completion of the studies with acid refractories.

Two views of the experimental converter are shown in fig. 6 and 7; sections through the converter are given in fig. 8; detailed data on the converter and its accessories are found in table II. The significant features are the following: (1) Air is blown across the surface of the metal rather than through it, (2) the space above the metal surface is enlarged appreciably over that of a standard side-blown converter and (3) the capacity may be increased by increasing the length along the cylindrical axis. Enlargement of the area above the bath should tend to reduce slopping and spitting encountered in side-blown operation by offering no lateral support for foaming slag, by reducing the velocity of the air, and by introducing into the air stream a swirling motion which would tend to deposit entrained particles on the side-walls of the vessel. Finally, the enlargement of the vessel by increasing its length and inserting more tuyeres should not affect the critical factors of bath depth, bath width, and amount of air blown per ton of metal.

Vessel Operation

The program for the operation of the experimental converter was set up and the work so arranged that commercial practices were followed as closely as practicable in order to give a comparison of the behavior of the experimental converter with that of regular vessels. Accordingly, the blown metal was deoxidized as in regular Bessemer practice and teemed into 18x18 in., 22x24 in. or 25x25 in. molds. The types of steel made were similar to those produced in the regular Bessemer converter including rimmed, mechanically capped, and silicon capped practices. The ingots were sent to the blooming mill, rolled and fabricated into commercial products. Samples were obtained from the billets for laboratory tests of physical properties.

Table II. Data on 3-Ton Experimental Converter and Accessories

1.	Vessel Dimensions (see fig. 8)	
	Height (outside)	15 ft 3 in.
	Diameter (outside)	12 ft
	Length (outside)	6 ft 6 in.
2.	Vessel Lining (see fig. 8)	
	Thickness	1 ft 2 in. to 2 ft 9 in.
	Material	Mica Schist
3.	Tuyeres	
	Number	6
	Size	1¼ in. id, 6 in. od, 34 in. long
	Material	Fireclay
4.	Bath Data (Estimated)	
	Size of Bath Max. Depth	Surface Dimensions
	3 Ton 8 in.	5 ft x 3 ft 8 in.
	5 Ton 12 in.	5 ft 6 in. x 3 ft 10 in.
5.	Mouth Opening (Estimated)	23 x 33 in. (with 8 in. radius on corners)
6.	Auxiliary Equipment	
	Drive Mechanism	35 HP Gear Motor
	Iron Ladle	7 ton capacity
	Steel Ladle	10 ton capacity
	Molds	18 x 18 in., 22 x 24 in., 25 x 25 in.
7.	Air and Steam Supply	
	Air Pressure in Line	24-28 psi
	Operating Air Pressure	5-15 psi
	Operating Air Volume (Flow)	3-10,000 cfm
	Size of Air Line	10 in.
	Air Shut Off	Manual
	Air Pressure Control	Manual and Automatic
	Air Volume Control	Manual and Automatic
	Size of Steam Line	2½ in.
	Maximum Steam Flow	120 ppm
	Steam Control	Manual
8.	Instruments	
	Air Pressure	Indicating and Recording
	Air Volume	Indicating and Recording
	Air Temperature	Indicating and Recording
	Steam Volume	Indicating and Recording
	Steam Pressure	Indicating and Recording
	Flame Record	Indicating and Recording
	Total Air Volume	Indicating
	Total Steam Volume	Indicating

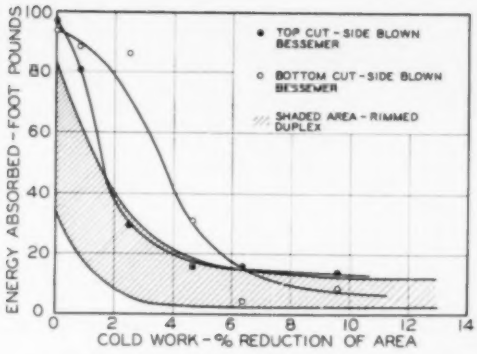
ENERGY ABSORBED - FOOT POUNDS
COLD WORK - % REDUCTION OF AREA
TOP CUT - SIDE BLOWN BESSEMER
BOTTOM CUT - SIDE BLOWN BESSEMER
SHADED AREA - RIMMED DUPLEX

Fig. 5—Relative work sensitivity of side-blown Bessemer and rimmed duplex steels.

The operation of the experimental converter was, of necessity, limited to one turn a day so that the converter, ladles and molds were cold at the start of each turn. About 41 pct of the 185 heats made were blown with a relatively cold vessel, whereas the commercial Bessemer converter operates at all times with an inside vessel temperature of well over 2000°F. Thus the problems of control were complicated by rapidly changing temperature conditions for the first heat of each day and it was difficult to maintain uniform pouring and teeming temperatures. Also, because of the location of the experimental unit, it was necessary to allow the ingots to cool in the molds and bank them until a sufficient number was accumulated to permit a rolling schedule. As a result, only about 33 pct of the heats can be considered satisfactory on all of these counts; yet 90 pct of all the heats were applied experimentally to commercial products. Under normal mill conditions, the uniformity of the steel would obviously be better.

Table III. Operating and Metallurgical Data On Three Types of Experimental Converter Blows

	Blow No. RB 103	Blow No. RB 121	Blow No. RB 115
Date Blown	3/19/43	4/6/43	4/2/43
Type of Blow	Surface	Subsurface	Combination
Charge X1000 lb	14.0	14.5	12.8
Blowing Time Min	22.5	20.0	18.0
Air Pres. psi	2-4	6.0	6.5
Air Flow 1000 cfm	5.5	5.0	5.0
Tuyere Size	$2^3/_4$ in.	2 in.	2 in.
Tuyere Angle	8°	8-1°	6.5-2-1°
Iron Composition			
	Pct	Pct	Pct
Manganese	0.34	0.44	0.50
Phosphorus	0.098	0.113	0.102
Sulphur	0.038	0.028	0.041
Silicon	1.17	1.30	1.28
Steel Composition			
	Pct	Pct	Pct
Carbon	0.12	0.08	0.08
Manganese	0.36	0.43	0.44
Phosphorus	0.106	0.112	0.105
Sulphur	0.038	0.025	0.034
Silicon	0.03	0.03	0.03
Nitrogen	0.003	0.007	0.006
Slag Composition			
	Pct	Pct	Pct
FeO	36.25	24.38	28.38
Fe_2O_3	2.30	1.43	1.43
Al_2O_3	3.05	2.68	2.01
MnO	3.16	9.30	11.16
CaO	0.81	0.73	0.20
SiO_2	53.78	60.40	56.20

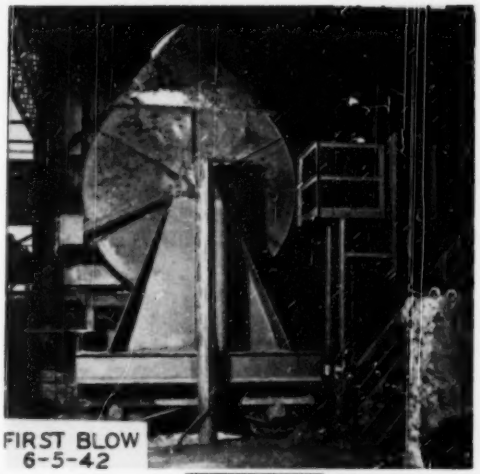

Fig. 6 (above)—Three-ton, J & L, side-blown converter (side view).

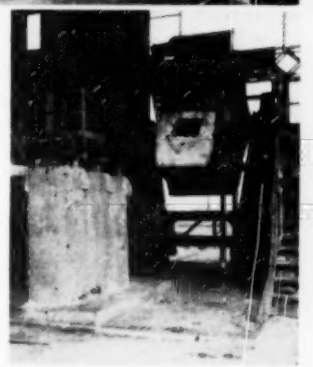

Fig. 7 (right)—General view of the converter shop.

Three distinct methods of operation were employed: (1) Surface-blowing, in which the tuyeres are located at or above the surface of the bath, (2) subsurface-blowing, with the tuyere openings below the metal surface, and (3) combination blowing, that is, use of both of these methods during a single heat. Fig. 9 illustrates the positions of the converter for surface and subsurface operation. Factors such as temperature and composition of the metal and slag, blowing time, fluidity of the slag, and vessel efficiency are all affected by the blowing method used. Table III is an excerpt from the operating data sheet and illustrates the type of information collected.

Experimental Results

In summarizing converter performance, the data have been divided so that surface blowing and subsurface blowing can be compared with each other and with bottom-blowing practice. Whenever possible, use is made of frequency diagrams to illustrate the behavior of the process under these two conditions of operation. Since combination blowing would lie between these extremes and would vary all the

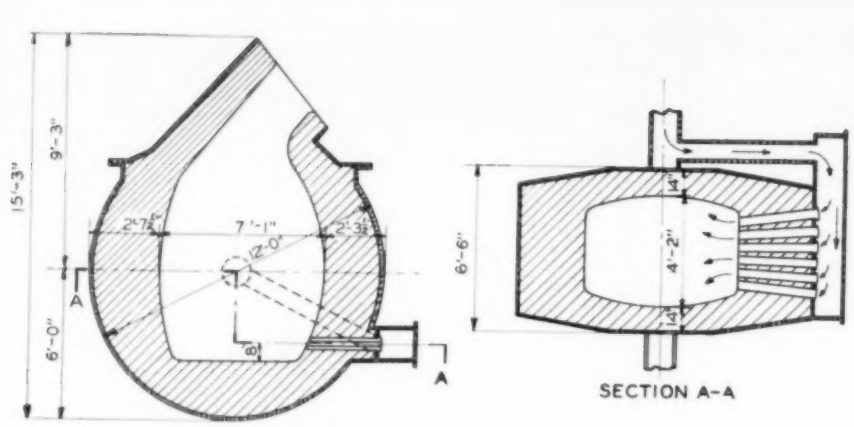

Fig. 8—Sectional views of J & L experimental converter.

way from one to the other, depending on the portion of the blow employing each method, it has not been included for comparison.

Steel Composition: In the majority of heats the steel composition desired was: 0.08 pct carbon, 0.30-0.40 pct manganese, low silicon, phosphorus and sulphur as dictated by iron composition and as low a nitrogen value as possible.

Nitrogen—The nitrogen content produced by subsurface operation was slightly higher than by surface blowing and both methods gave a large reduction over that of the bottom-blown mill converters. Fig. 10 shows the distribution of nitrogen content of the heats made by each method of blowing and for comparison, nitrogen ranges of other steelmaking practices also are given. It will be noted that the surface-blown heats fall well within the range for scrap practice open-hearth steel.

Carbon—Fig. 11 gives the distribution of carbon content in the finished steel for surface and subsurface blowing. It is evident from these curves that surface blowing does not produce as low carbon as regular bottom blowing Bessemer though, by subsurface operation, carbon values closely approaching the bottom-blown converter can be obtained. Where low carbon is desired along with the lowest nitrogen content, the combination blow, consisting of surface operation for most of the period and subsurface blowing at the finish, appears to be the logical choice.

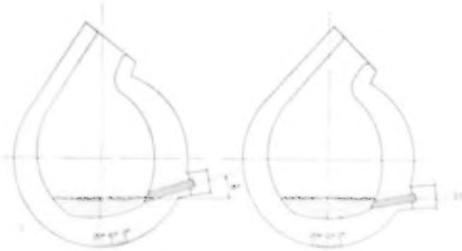

Fig. 9—Converter positions for surface and subsurface blowing.

Table IV. Typical Compositions of Converter Slags

	Pct FeO	Pct Fe_2O_3	Pct MnO	Pct SiO_2	Pct Al_2O_3
Experimental Surface-blown	38.18	3.43	3.81	49.10	3.18
Experimental Subsurface-blown	27.73	1.50	8.56	58.50	2.34
Regular Bottom-blown	16.32	2.91	10.00	67.61	2.12

Table V. Range of Temperatures for Experimental Steel

(Optical readings on the stream)

	Maximum °F	Minimum °F	Average °F
Surface	3157	2840	3006
Subsurface	3064	2880	2998

Manganese—The manganese content is largely a function of the efficiency of the manganese addition to the ladle. With surface-blown heats the efficiency varied considerably because a thin converter slag was produced by this process and varying amounts of the slag reached the ladle where it reacted with the manganese addition. In the case of subsurface blowing the slags were much thicker and, in the main, were held in the converter. Under these conditions the manganese content remained uniform and the efficiency of the added manganese was normal (approximately 72 pct average).

Other Elements—Phosphorus is not removed in an acid lined converter, therefore the phosphorus content was determined by the iron composition. Silicon removal by subsurface operation was comparable to the regular bottom-blown converter but surface blowing gave slightly higher silicon values. Unlike acid bottom-blown operation, sulphur was removed by subsurface blowing to the extent of 14.5 pct average and by surface blowing, 16.5 pct average. The mechanism by which this is accomplished is not clearly understood.

Slag Compositions: Surface blowing gave thin, fluid slags that drained into the ladle and interfered with ladle deoxidation. The slags produced by sub-

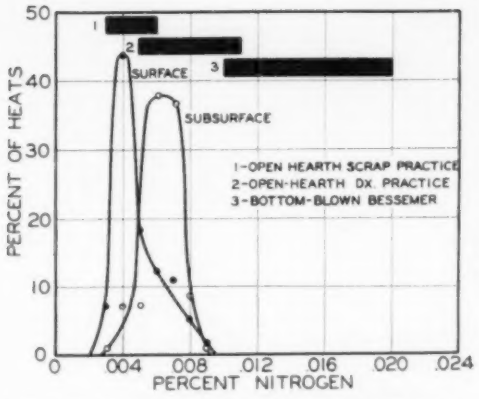

Fig. 10—Frequency distribution of the nitrogen content of the finished steel for surface-blown (109 blows) and subsurface-blown heats (57 blows).

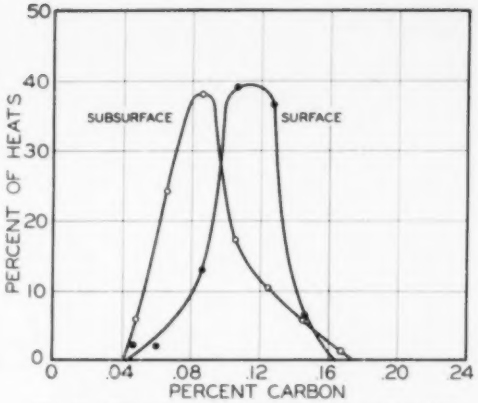

Fig. 11—Frequency distribution of the carbon content of the finished steel for surface-blown (109 blows) and subsurface-blown heats (58 blows).

Table VI. Bath Depths and Average Blowing Times for Various Sized Heats

Size of Charge, Lb	Bath Depth, In.
6,000	8
10,000	12
16,000	21

Size of Charge, Lb	No. of Heats	Avg. Blowing Time, Minutes
6,000-10,000	46	14
10,000-16,000	33	19
16,000-22,600	10	30

surface blowing were drier and handled more like slags from the bottom-blown converter. Table IV gives examples of typical slags obtained by surface, subsurface, and regular bottom-blowing practices. The most significant change is in the FeO content which has a direct effect on the fluidity of the slag.

A decided decrease in the amount of slag produced by the subsurface-blowing method was observed. The majority of heats had approximately 125 lb of slag per ton of iron which is comparable to the weight of slags from regular bottom-blown converters and is less than that from surface-blown heats. This is attributed to the more violent turbulence throughout the bath during subsurface blowing in place of the limited metal and slag agitation that occurs during surface blowing.

Temperature: The temperatures obtained during subsurface blowing were slightly lower than those on surface-blown heats. Table V shows the maximum, minimum, and average temperatures for both surface and subsurface-blown heats. While the averages are fairly close, the range of temperatures for surface-blown heats is greater than that for subsurface-blown heats. The narrow range of temperature experienced with the latter practice was a factor in securing better control of the slag and greater manganese efficiency.

Size of Charge: The size of the heats made during the campaign varied from 6000 to 22,600 lb. Originally the vessel was designed for approximately 6000-lb charges. During the early operations, some difficulty was experienced in handling the relatively small amounts of metal and subsequently, the charge was increased to 13,000 lb which gave reasonably uniform operation of the vessel and permitted pouring an ingot of mill size (22x24 in.). Later, in an effort to determine the maximum operating capacity of the shell, charges containing up to 22,600 lb of iron were handled. This is considerably in excess of the optimum capacity for the vessel and to accommodate a charge of this size it was necessary to remove part of the lining in the bottom of the converter. Table VI shows the bath depth and blowing times for several sizes of charges. Since the air flow could not be increased proportionately with the size of charge, the time of the blow was longer for the larger heats.

Blowing Time: Blowing times in minutes per ton for subsurface-blown heats were less than for surface blows as shown by the distribution curves of

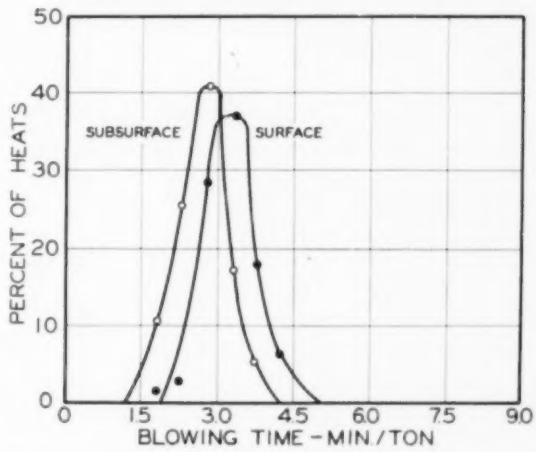

Fig. 12—Frequency distribution of blowing time (minutes per ton) for surface-blown (109 blows) and subsurface-blown heats (58 blows).

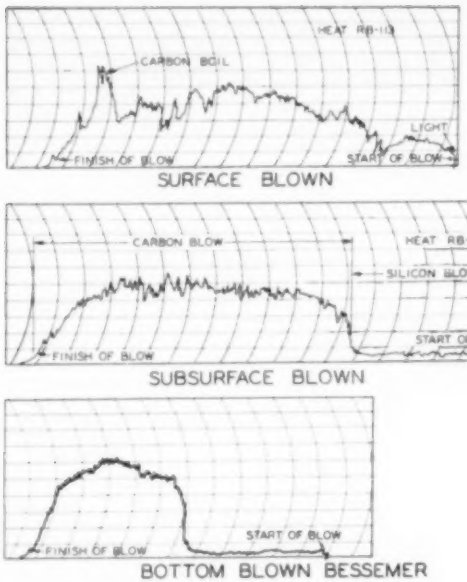

Fig. 13—Photocell records of converter flames obtained by surface, subsurface and regular bottom blown Bessemer operation.

fig. 12. The total time of blow was a function of the rate of air flow which, in turn, was limited by the tendency for the converter to slop. The faster rate of blowing by subsurface operation can be attributed partially to the greater air penetration with its resultant higher reaction efficiency and to the effect of the metal in breaking up the air jets and reducing the air velocity.

Air Flow and Pressure: Air pressures required in the experimental converter were relatively low as compared to the regular bottom-blowing vessels. Under all the various conditions of operation, operating air pressure ranged from 2 to 17 psi and with an air flow of 2000 to 9000 cfm. An air flow of 5500 cfm was found to give optimum converter performance for all sized charges used and the air pressure necessary to give this flow was approximately 6.5–7 psi.

Control of the Side-Blown Process: The flame from the side-blown converter is different in appearance from that of the bottom-blown converter and therefore the control of the blow could not be based closely on this practice. The most marked difference in the photocell records shown in fig. 13 will be noted in the case of surface blowing while the subsurface blow occupies an intermediate position. There appear to be a number of types of flame during the carbon elimination period of the surface-blow, and the one encountered for a particular blow will depend upon the metal temperature. The first, corresponding to a cold blow, is an almost transparent flame of yellow-green color with very little smoke, similar in appearance to that of a regular Bessemer blow that is being steamed. Through blower's glasses, it appears as a bright green. The steel made by such a blow would be too cold to be successfully teemed—at least for the small heats made in the pilot unit. At a slightly higher temperature the flame changes to dull red and quantities of smoke are emitted. As the temperature continues to increase, the flame brightens until, through blower's glasses, green streaks begin to appear at the edges. The body of the flame still has a red color that becomes more intense as the temperature rises. A heat in which the green streaks appear about midway in the carbon blow is hot enough for proper handling. With further increase in temperature, the interior of the flame changes to a lighter shade of red and the number and size of the green streaks increase. At still higher temperatures the edges begin to break up and fan out or "feather." When the flame is covered with these feathers and the green color has spread to

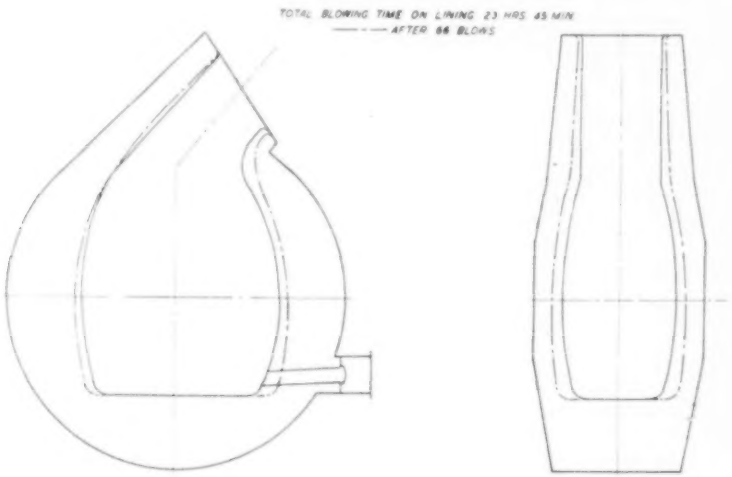

Fig. 14—Lining wear in experimental converter. Original lining.

Solid line—original lining.
Broken line—after 66 blows.

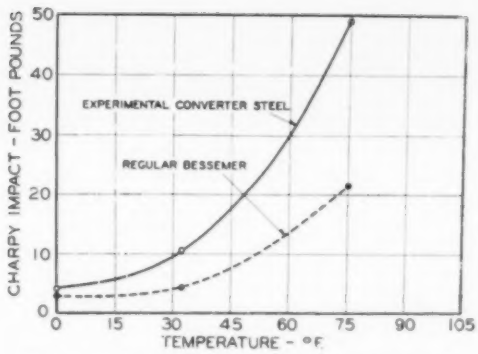

Fig. 15—Embrittlement properties of experimental converter and regular Bessemer steels at varying temperatures.

include almost all the flame, the blow is considered to be very hot.

The sparks from the converter are likewise of some assistance in judging the temperature since their color becomes brighter with increase in temperature. If observations are made on both the flame and sparks, the temperature can usually be judged with fair accuracy. Under standardized blowing conditions, where the air flow, tuyere size and charge are fixed, the flame intensity as shown by a recording photocell unit provides a guide to blowing temperature.

The end-point of the blow can be determined both by photocell equipment and by viewing the flame. The bottom-blown converter endpoint occurs at approximately 1/3 to 1/2 the peak intensity of the carbon flame as shown by the electric eye, and at a much smaller change as judged by the eye of the observer. In the case of subsurface blows the flame decreases to about 1/10 its original intensity as seen by both the photocell and the observer. With blower's glasses, the color shifts to a darker red with the green fringes disappearing entirely at the finish of the blow. In the surface blow the endpoint comes rather suddenly and the flame intensity drops to a very low value. Here again the endpoint is noted by a disappearance of the green color in the flame.

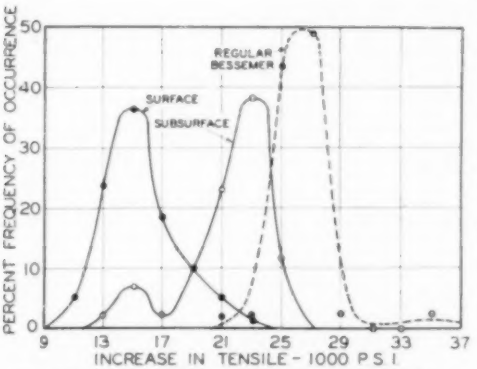

Fig. 17—Effect of nitrogen on the aging characteristics of regular Bessemer and experimental steels by the "blue heat" tensile test.

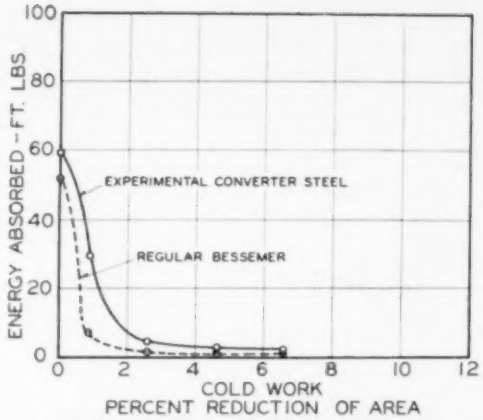

Fig. 16—Average work-brittleness results on experimental converter and regular Bessemer steels.

Tuyeres: Six single-hole tuyeres were used in the converter. Tuyeres made of both fire clay and metal pipe and with hole diameters from $1\frac{3}{4}$ to $2\frac{3}{4}$ in. were tested; the clay tuyere with a 2 in. diam hole was chosen as most satisfactory. The tuyeres were set in the converter at various angles depending on the operation; fig. 8 shows the tuyeres installed horizontally. However during subsurface blowing it was desirable to have the tuyere slope down into the metal bath so that iron could not run out the back and therefore the tuyeres were mounted at angles from 0° to 7° to the horizontal plane. The position chosen was governed by the size charge to be blown.

Vessel Refractory: Mica-schist stone and ganister mud were used as the acid refractories for lining the experimental converter. These are the same materials employed in the mill vessels but the method of lining is different in several respects. Since the bottom is an integral part of the experimental unit, the vessel was lined in the blowing position instead of being inverted as is done with the mill unit. The tuyeres were laid in place as the

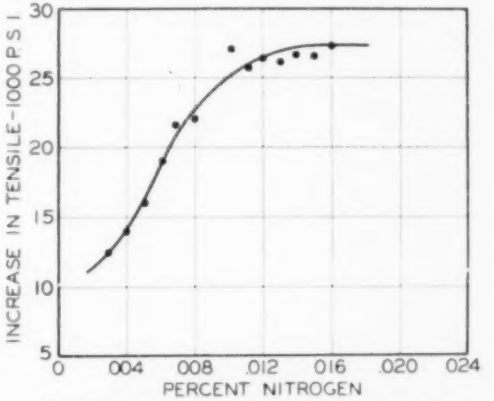

Fig. 18—The effect of nitrogen on aging as shown by "blue heat" tensile tests of experimental converter and bottom-blown converter steels.

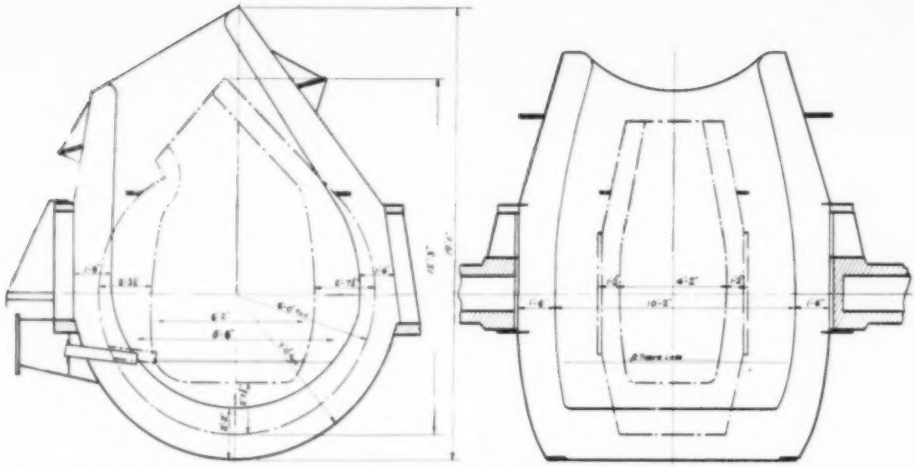

Fig. 19—Comparison of the J & L 3-ton and 22-ton side-blown converter contours.

Table VII. Laboratory Data On Three Types of Experimental Converter Steel

	Blow No. RB 103	Blow No. RB 121	Blow No. RB 115
Heat Treatment	Normalized	Normalized	Normalized
Application	Wire	Pipe	Wire
Check Analyses			
	Pct	Pct	Pct
Carbon	0.11	0.08	0.05
Manganese	0.37	0.43	0.45
Phosphorus	0.108	0.118	0.117
Sulphur	0.041	0.031	0.033
Silicon	0.01	0.02	0.01
Nitrogen	0.003	0.007	0.005
Physical Properties at Room Temperature			
Yield Point—Psi	44,940	38,750	39,000
Ultimate Tensile Strength—Psi	63,160	60,750	60,150
Percent Elongation	35	36	33
Percent Reduction of Area	67	69	68
Physical Properties at 500°F			
Ultimate Tensile Strength—Psi	75,750	81,710	82,250
Increase—Psi	12,590	20,960	22,100
Work Brittleness Test Results			
Cold Work, Pct	Izod-Foot Pounds		
0	59.5	38	58.5
0.7	32	5	16
2.6	4	1	2
4.5	3	2	2
6.3	4	2	2
8.7	4	2	2

lining was constructed so that they became a part of the lining. No provision was made for rapid change of the tuyeres in the original design and later it was determined that none was needed.

With the first lining, 66 blows were made before a major patch was required and, of these, 27 blows or 41 pct were first blows of the day when spalling would be most serious. Since the converter was operated on a one-turn-per-day basis, a gas flame

Table VIII. Applications of Experimental Bessemer Ingots

Application	No. of Heats	Percent of Total
Common Wire	75	40.6
Conduit	28	15.1
Fine Wire	21	11.4
Tie Plate	12	6.5
Pipe	3	1.6
Screw Stock	1	0.5
Not Applied	45	24.3
Total	185	100.0

was used to heat the vessel overnight. This is recognized as being not entirely satisfactory since the temperature of the vessel lining never exceeded 1500°F for the first blow, but was most convenient under the conditions of operation. The experimental vessel lining underwent therefore, a severe thermal shock on 41 pct of the heats while a mill converter is subjected to this shock only on the first blow of a new lining or patch. A drawing of the original lining, showing the inside contour after 66 blows, is presented in fig. 14. It will be noted that the wear was fairly uniform around the walls of the vessel with the greatest wear occurring at the shoulder in the back wall near the nose. This shoulder caused some difficulty throughout the investigation in that all linings showed considerable loss at this point and it was not easy to attach a patch. The tuyeres wore at approximately the same rate as the lining so that it was necessary to change them only when the lining was being repaired.

Physical Properties of Experimental Steel

The composition of the experimental steel, insofar as the common elements are concerned, was similar to regular Bessemer steel and interest in its physical properties centered around those known to be influenced by nitrogen content.

Laboratory tests indicate that the experimental steel is superior to the regular Bessemer product for applications involving cold work. It has comparable tensile and yield strengths and somewhat better elongation and reduction of area; the impact

strength is higher than regular Bessemer steel at room temperature but as the temperature is reduced to 0°F the impact strength of both steels is low. Work-brittleness tests indicate that the experimental steel is somewhat less sensitive to cold work and "blue heat" tensile tests show that its aging characteristics are considerably better than those of bottom-blown converter steel.

Examples of laboratory results on the physical properties of the experimental converter steel are given in table VII and curves summarizing data on its sensitivity to low temperature embrittlement, cold work and aging are shown in fig. 15, 16, 17, and 18. The similarity of the curve in fig. 18, which was made using experimental converter heats and regular Bessemer heats, and fig. 3 made from data on laboratory induction furnace heats, is marked.

Application of Experimental Steel to Commercial Products

Whenever possible, the steel was applied experimentally to commercial products and table VIII shows the disposition of the heats. Applications were generally selected on the basis that cold working was an important factor, and that differences in properties could be readily determined. Wire and conduit seemed to offer the most promising possibilities and the majority of the heats were thus applied. To provide control data, companion heats of regular Bessemer steel were also tested so that direct comparisons could be drawn. Coincident with this work some difficulties were experienced in the application of regular Bessemer steel to tie-plate and to a screw-stock application involving cold work, and experimental converter steel was used successfully.

The force needed to bend pipe is an important factor in the fabrication of such material and provides a simple measure of the cold forming properties of the steel. Table IX gives the bending force under standardized conditions for ½-in. diam standard pipe made of experimental converter steel, regular Bessemer and rephosphorized open-hearth steel. Similar tests were made on ½-in. conduit, ¾- and 1-in. pipe and the grading of the steels was comparable. As was expected, the experimental converter steel was intermediate between the Bessemer and the high phosphorus (0.050 pct) open-hearth steels. The applications of the experimental converter steel to other products such as fine wire, tie plate and screw stock also gave results indicating that this steel occupies a place between that of Bessemer and open-hearth steels in cold working behavior. Further improvement in the product would require a lower phosphorus content.

Design and Operation of the 22-ton Unit

Upon completion of the work with the 3-ton converter, it was decided to build a large mill converter for full-scale experiments on blowing time, lining life, blown metal composition, yields, etc. Since this vessel could be operated most conveniently in the existing mill stands, certain compromises in design had to be made. The spacing of trunnion supports in the mill stand dictated the vessel length, and the interior cross-section from the tuyeres to the opposite side of the new converter was made somewhat larger than that of the 3-ton unit so as to obtain a design charge of 22-tons. The difference in size and contour of the two vessels is indicated in fig. 19 while the converter shell is shown in fig. 20 and 21.

Table IX. Test Data on ½ in. Standard Pipe—Average Values

Type of Steel	Y.P. Psi	U.T.S. Psi	Pct Elong. in 6 In.	Bend Test In.-Lb
Experimental Converter	48,790	60,823	29.1	2455
Regular Bessemer	50,705	67,310	25.6	2670
Hi-Phos. (0.050 pct) Open-Hearth	42,380	55,140	30.0	2350

The experimental 22-ton side-blown converter was set up in No. 1 converter stand at the J&L Aliquippa Works and was operated by the Aliquippa Bessemer Department from Dec. 14 to 19, 1946. By using different blowing practices and several compositions of iron, a considerable amount of information on the performance of the converter was obtained. Table X summarizes the operation.

Nitrogen Content of Blown Metal

Primary objectives of this development were the production of low nitrogen blown metal for duplexing, and of Bessemer steel with superior cold working properties. The nitrogen content ranged from 0.003 to 0.008 pct with an average content of 0.0049 pct for the 29 blows. These values are comparable with those obtained in the small vessel.

Converter Capacity

The converter unit was designed to blow a maximum of 22 tons in not over 20 min. In actual operation, it produced 22 tons in as short a time as 10½ min and approximately 34 tons in a minimum time of 17 min. There is no evidence that the maximum charge of the converter was reached, and larger charges probably could have been blown successfully with longer blowing times.

Control of the Process

The test showed the major control problems of the process to be (1) blown metal temperature, (2) slag consistency and (3) carbon content of the blown metal.

Metal temperatures tended to be considerably higher with the side-blown converter than with the bottom-blown unit. An effort was made to keep temperatures normal by the use of steam, scrap and ore: scrap charges as high as 21 pct of the hot metal charge were used and the temperature was still higher than normal. Properly exploited, this feature of the converter could be valuable either as a low cost means of melting scrap or to obtain direct conversion of iron ore to steel.

Slags were generally quite fluid—caused in part by the high operating temperatures—and, under conditions of the test, it was not possible to separate the slag from the metal. The fluid slag interfered with deoxidation, and manganese in the finished steel was too low to make it comparable with any normal Bessemer composition. It should be noted that these results were less satisfactory than were obtained in the small vessel. It is recognized that this is an important problem and in any future trials, control of the slag should be investigated along the following lines: (1) use of skimming devices, or special flush techniques, (2) better temperature regulation, (3) additions to slag, and (4) modification of vessel design.

Experience with the original 3-ton side-blown converter had indicated that flame appearance and,

Table X. Side Blown Bessemer Data Sheet

Blow No.	Total Charge Wt. Lb	Scrap Wt. Lb	Finish Wt.* Lb	Blowing Time Min	Blowing Time Sec	Method	Operation	Blast Pressure Psi	Composition †	C	Mn	Si	N_2
1	43,000			10	30	Subsurface	OK	21-29	B	0.09	0.01	0.005	0.005
2	43,000			9	50	Subsurface	Slight Spitting	23-23	B	0.37	0.04	0.010	0.008
3	50,000	4000		12	55	Surface	OK	14-21	B	0.08	0.04	0.070	0.003
4	50,000	5000	40,000	13	0	Subsurface	OK	17-20	D	0.11	0.32	0.005	0.004
5	50,000			14	10	Subsurface	OK	17-21	B	0.06	0.08	0.04	0.004
6	48,000	3000		14	47	Subsurface	OK	15-23	B	0.06	0.08	0.05	0.004
7	50,000			13	8	Subsurface	Slight Spitting	17-21	B	1.49	0.09	0.13	0.005
8	50,000	4000		12	0	Subsurface	OK	18-22	B	1.12	0.09	0.09	0.005
9	54,000	4000		12	35	Subsurface	Slight Spitting	18-24	B	1.88	0.09	0.05	0.005
10	56,000	4000		12	15	Subsurface	OK	19-20	B	1.70	0.09	0.05	0.004
11	50,000	5000	41,000	14	20	Subsurface	OK	17-20	D	0.12	0.44		0.005
12	51,000			13	0	Surface	Slight Spitting	17-23	B	2.45	0.08	0.21	0.003
13	55,500	5500	52,500	14	50	Combination	Spitting	16-18	D	1.52	0.27		
14	49,000	4000		10	20	Surface	Spitting	11½-16	B	3.31	0.03	0.07	0.003
15	50,000			12	43	Surface	Severe Spitting	14-16	B	3.53	0.14		0.003
16	50,000	5000		15	11	Surface	Severe Spitting	13-15					
17	56,000	None		20		Combination	Slight Spitting	12.5-10.0	B	0.84	0.03	0.02	0.003
18	56,000	6000		25	30	Combination	OK	13.0-11.5-8.0	B	0.05	0.04	0.02	0.003
19	53,500	5500		20	45	Combination	OK	16-21	B	0.07	0.01	0.04	0.003
20	61,000	7000	54,000 +	23	10	Combination	Slight Spitting	14-20	D	0.14	0.21		0.005
21	61,000	7000	54,000	23	40	Combination	Slight Spitting	16-21	D	0.11	0.20		0.008
													0.007
22	64,000	7000	54,000	26	2	Combination	Very Severe Spitting	15-21	D	0.11	0.21		0.008
													0.008
23	62,000	7000	54,000	23	2	Combination	Slight Spitting	17-22	D	0.12	0.13		0.007
		3500 Butts					Slight Spitting	18					0.008
24	59,000	+ SiO_2 Bats	54,300	22	0	Combination	Spitting	19	D	0.07	0.18		0.008
25	58,000	6000 Ore	53,500	17	0	Combination	Slight Spitting	17½	D	0.08	0.12		0.005
26	60,000	9000 Butts	47,000	22	10	Combination	Slight Spitting	20	D	0.15	1.08		0.005
27	63,000	11,000	54,000	18	20	Combination	Slight Spitting	20-25	D	0.10	0.20		0.005
28	67,000	8000		16	50	Combination	Tuyere Section	16-18	B	0.11	0.08	0.08	
29	61,000	+ SiO_2 Bats		26		Combination	Burned Out		D	0.10	0.74		0.006

therefore, carbon control would be considerably different from that of the standard Bessemer converter and this was borne out in the 22-ton experimental converter. Several blows were made before the end-point could be definitely established. Later, the blowing technique was improved until the regular Bessemer photoelectric control unit could be used in end-point determination. As is normal with side-blown operation, the carbon finished 0.04 to 0.05 pct higher than in bottom-blown metal. By means of subsurface and combination blowing, the carbon content was brought to within a point or two of regular Bessemer metal.

Efficiency of the Process

Gross metallic yield varied considerably for the blows on which accurate data could be obtained, ranging from 78 to 92 pct. Part of the fluctuation in efficiency was due to changes in the blowing methods used. Under some conditions, yields comparable to those in bottom-blown converter operation were obtained but extensive continuous operation would be necessary to develop an optimum blowing practice and to determine the ultimate efficiency of the process.

Tuyeres and Lining

The tuyere design was somewhat different from that used in the 3-ton side-blown vessel and, as it turned out, was not nearly as satisfactory. The main fault was leakage past the head of the tuyeres and, though a false tuyere plate was used to bleed away the leaking air, some air seeped into the lining over the tuyeres. The air loosened the lining and produced deep holes directly above the tuyeres causing early failure of this section of the lining. To keep the vessel in operation, frequent patching was necessary and finally the lining opened up so that, in the 29th blow, molten metal reached the false tuyere plate. Since it had already been decided that most of the information that could be developed by this test had been obtained, no further attempt was made to patch the vessel and it was removed from the stand. A new tuyere arrangement has now been designed to eliminate the possibility of high pressure air entering the lining.

As in the regular Bessemer converter, mica schist and ganister mud were the lining materials. Excluding the sections above the tuyeres and at the nose, the lining at the completion of the test still approximated rather closely the original thickness. The nose wear was more severe and in some sections of the nose as much as 6 in. of the lining was lost. Whether this was due to the high operating temperatures or is to be expected in normal operation can be answered only by further work.

Evaluation of the Experimental Work

In comparing the performance of the two converters, it will be noted that in several particulars the 22-ton converter bore out the findings of the 3-ton unit while in a few cases the characteristics of the large converter were not as satisfactory as those of the small vessel.

Blown Metal: The blown metal made by the two converters under similar practices was comparable in composition. Surface blown heats gave higher carbon contents while subsurface and combination blowing produced steel with carbons approaching those of the bottom-blown process. What effect the higher carbon might have on the application of the steel to products is unanswered at present but no ill effects were apparent in the use of the experimental steel in furnace welded pipe, a product for which low carbon steel is specified. Of course, for duplexing operations, the carbon in the blown metal would be advantageous.

The nitrogen content of the steel from both converters was low and, in a considerable proportion of the heats, compared favorably with that of scrap-practice open-hearth steel. In this respect each converter fulfilled the expectations of the investigation.

Process Efficiency: Both converters were capable of producing very high temperature blown metal and this would appear to constitute a primary ad-

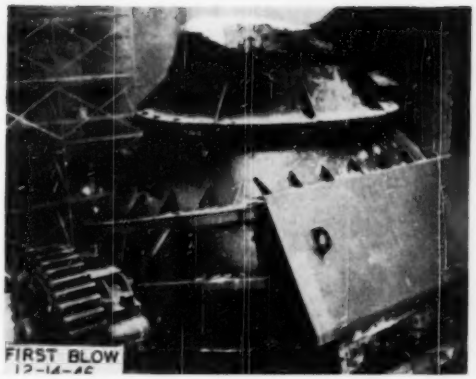

Fig. 20 (above)—View of 22-ton side-blown converter showing wind-box and row of tuyere openings at the right.

Fig. 21 (right)—View from front of 22-ton side-blown converter.

vantage of the process since the extra heat could be used either to melt additional cold scrap or, using lump ore as a substitute, produce metal by direct reduction.

Metal yield of the converters varied rather widely but in each case the best yields compared favorably with bottom-blown practice. The principal losses were due to spitting during the blow and to the higher iron oxide content of the slag. Experience thus far indicates that under the best operating conditions the iron oxide content of the slag likely will be higher than in bottom-blown operation. Some advantage may be gained however by the lack of spitting loss during the turn-up and turn-down of the side-blown converter.

With the fluid slags obtained by side-blowing, the life of the lining will undoubtedly be less than in a Bessemer converter. This, however, may be more than offset by the longer tuyere life to be expected from the side-blown vessel. It is unfortunate that this fact could not be demonstrated in the second test due to the trouble with air leakage mentioned above. The tuyeres in the 3-ton experimental vessel wore back at the same rate as the lining itself when no air leaks through the lining were present. Thus, it appears that a replaceable tuyere section, comparable with the short-lived bottom in a regular Bessemer converter, may not be required; the tuyeres would be replaced only when the vessel is relined.

The capacity of the 22-ton converter was not obtained solely by lengthening the cylindrical section of the 3-ton converter; the vertical cross-section was increased somewhat and the metal bath deepened so that greater air flow per unit area of bath surface was necessary to maintain comparable blowing times. The increased air rate could be obtained only by higher air velocity with an attendant step-up of losses by entrainment of material in the air stream.

Another result of the larger cross-section was an increase in the distance from the tuyere opening to the opposite side of the metal bath. Experience with foundry converters has shown that when this distance becomes too large the slags have a tendency to increase in iron oxide content and be thin or watery. By keeping this dimension small, the slag and metal are maintained in violent agitation by the air so that the iron oxide content of the slag and metal are held close to equilibrium by the reducing action of the carbon in the metal bath. When the dimension from the tuyere to the opposite side of the bath becomes large, the slag is blown to the far side where, with the air velocity much lower, a relatively thick layer builds up and is no longer mixed with the metal. Since the slags of the 22-ton converter were thinner than those of the 3-ton converter, consideration should be given to a study of cross-section dimensions in any further work on the process.

At the close of the war it was discovered that the Germans at the Mannesmann Works had been experimenting with a basic lined side-blown converter of commercial size.[9] This was a regular 30-ton bottom-blown converter with a blank bottom and a tuyere block consisting of seven rows of holes fitted into the side. Apparently this converter produced steel having a low nitrogen content and improved cold working properties. Unfortunately, little operating information is available on the converter so that full advantage cannot be taken of this work in furthering the design of a commercial unit.

First consideration was given to the adaptation of the Peirce-Smith copper converter to this steelmaking process. The converter is somewhat similar in appearance to the present experimental unit and is made in various capacities up to and exceeding 100 tons. As it has been in use for many years in the copper industry, its design is well standardized and it is relatively inexpensive. Its 13-ft diam cross-

section is intermediate between that of the two experimental units so that modification would be held to a minimum.

Several other designs have been suggested. Inherent in these has been the concept of a long horizontal cylinder with the tuyere constants, the depth of bath and the sweep of air across the bath established for optimum vessel operation. The length of such a cylindrical vessel could be adjusted for wide ranges of capacity and blowing time without affecting these critical factors.

Conclusions

The experimental work thus far has permitted a fairly comprehensive evaluation of the metallurgical advantages of the side-blow converter. Some of the operating characteristics of the converter have been determined; there remains to be solved a number of complex operating problems before the method can be considered a commercial success. The specific information developed in the course of the work with the two experimental vessels is outlined below:

1. Low nitrogen steel is produced within wide limits of operation of the side-blown converter.

2. Carbon content is higher than with the bottom-blown converter by 0.01 to 0.05 pct; this is definitely a function of the method of blowing.

3. High temperatures are attainable, the highest temperature being associated with surface blowing.

4. Since the thermal efficiency is high, larger amounts of scrap or ore may be used in the charge.

5. There appears to be no inherent limitation on size of charge, weights of iron from 3 to 30 tons having been blown successfully.

6. There is no major change in operation of a given side-blown vessel with fairly wide changes in size of charge; the small converter designed for 3 tons handled over 11 tons of iron successfully.

7. Slags were generally higher in iron oxides and consequently less viscous, but considerable control was obtained by variations in blowing practice.

8. Blowing losses varied widely for different operating conditions during the tests. At the best, these losses were comparable with standard bottom-blowing practice.

9. Lining wear was more rapid with side-blown operation than in bottom-blown practice due primarily, it is believed, to higher operating temperatures and thinner slags.

10. Premature failure of tuyeres and adjacent lining in the large vessel seems to have been the result of the specific tuyere design since similar difficulties were not encountered with the small vessel.

11. Much of the steel from the small converter was of commercial quality; there was no difficulty experienced in making butt-weld pipe, wire, tie-plate, and screw stock.

12. The cold working and aging properties of this steel were typical of a low nitrogen, high phosphorus steel. It was superior to regular Bessemer steel on cold forming applications.

Cooperative Work on Basic Converter: Converter steel is made commercially in both basic and acid lined vessels. The investigations described in this paper dealt with acid linings in a small experimental and a large plant-size side-blown converter. Independently, Carnegie-Illinois had carried out work on basic linings for converters of this type. On completion of the work with the acid lining in the 22-ton converter at Aliquippa, the project was made a cooperative one and the converter was lined with basic refractories and a series of heats made at the Carnegie-Illinois South Chicago Works. The studies with basic linings are described in a concurrent paper.

Acknowledgments

An investigation of this magnitude could have been carried out only with the active cooperation of many people in the operating, engineering and metallurgical departments. The writers wish to acknowledge the part played by all of these men and by their associates in the research laboratory, and to thank the management of Jones & Laughlin for permission to publish the results.

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Preferred Orientation as a Factor in

Intergranular Corrosion

by W. D. Robertson

INTERGRANULAR corrosion of metals and alloys has been extensively investigated and the current conclusions are summarized in the Symposium on Stress-Corrosion Cracking of Metals.[1] It appears, however, that the relative importance of the various factors involved in the mechanism is still obscure.

Several recent, independent studies, in combination with available X ray data on preferred orientation, indicate that there may be a direct correlation between relative orientation of contiguous grains and susceptibility to intergranular corrosion. The object of the present note is to show the nature

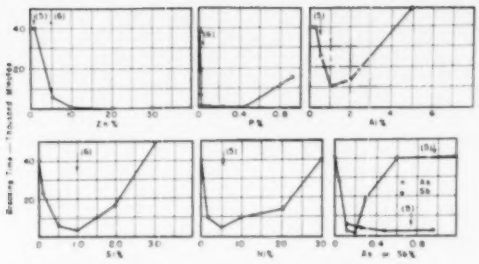

Fig. 1—Breaking time of recrystallized copper alloys in ammonia atmosphere at 10,000 psi stress (reproduced from Thompson and Tracy) compared with the alloy concentration producing a random recrystallization texture (arrows).

of the correlation and thereby to demonstrate that a few critical experiments are indicated by means of which the role of structure can be further clarified and quantitatively established.

Lacombe and Beaujard[2] have shown that the boundary of a single grain embedded in another grain of different orientation is attacked more or less rapidly depending on the relative orientation of the two adjacent crystals. An exactly similar effect is demonstrated by Dunn and Lionetti[3] with three-grain specimens of silicon ferrite of predetermined orientation having the (110) plane in the surface of the aggregate. The boundary associated with the largest difference in orientation (15°) was more attacked by the etching reagent than either of the other two boundaries where orientation differences were respectively 7 and 8 degrees.

Copper is ideal for the study of orientation dependent attack because it can be produced either in a random or a highly oriented state depending on the conditions of working and recrystallization. In addition, certain critical concentrations of alloying elements inhibit the formation of the oriented texture in straight rolled and recrystallized binary alloys. Comparison of the alloy concentration at which random orientation is produced with the concentration at which the alloys become susceptible to stress corrosion cracking indicates that the two effects are related.

Fig. 1 is a reproduction of curves from the work of Thompson and Tracy[4] showing time to failure in ammonia vapor at constant applied load for seven binary copper alloys; the limits, as far as they are known from X ray studies,[5,6] at which the texture of pure recrystallized copper changes to a more or less random orientation are indicated by the arrows.

Considering the heterogeneous character of the data and the range of values, correlation appears excellent, particularly in the case of zinc, phosphorus and aluminum; in the remaining cases data are lacking for concentrations below those indicated by the arrows but it is certain that the oriented texture is suppressed at lower alloy contents. Iron-nickel alloys may also be produced in cube-texture form and the results of Snoek,[7] previously noted by Lacombe and Beaujard, show that the oriented structure is immune to grain boundary oxidation at high temperatures, whereas the randomly oriented alloy is subject to intense intergranular attack.

It seems probable that as angular deviation between the lattices of adjacent grains increases, the chemical activity of the grain boundary zone will also increase, resulting in grain boundary attack. When stress is applied to a polycrystalline aggregate the elastic anisotropy of individual crystals could set up stresses at the boundaries, depending on the orientation. It is necessary, therefore, to determine the relative importance of the inherent chemical activity of the grain boundary, depending on orientation, and the effect of stress concentration at grain boundaries operating through elastic anisotropy to produce the acceleration of corrosion associated with stress-corrosion cracking. The effect of various alloying elements may be related either with the chemical or structural properties of the boundary, possibly both.

In view of the preceding correlation, a study of the chemical and mechanical properties of the grain boundary as a function of the orientation of adjacent grains appears to offer the best possibility of determining the mechanism that accounts for the various known phenomena.

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Technical Note No. 31 E. Manuscript received Sept. 23, 1949.

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Modern Baghouse Practice

for the Recovery of

Metallurgical Fumes

by A. L. Labbe and J. J. Donoso

Hard-won experience in the operation of smelting plants has pointed the way to the most efficient design and economical operation of a baghouse system for recovery of metallurgical fumes. This paper treats of the principles governing bag filtration and the factors most directly affecting bag life. Methods for temperature conditioning of smoke previous to filtering are described and the evolution of bag shaking mechanisms is brought up to date.

THE history of the bag filter process, as applied to the recovery of metallurgical fumes, goes back to 1876, when the Lone Elms Works of Joplin, Missouri, erected a two-section baghouse for filtering fumes from Scotch Hearth lead smelting furnaces. By modern standards this installation was crude, but the basic operating principles remain essentially the same today. Great progress, however, has been made in the mechanization of the process by the use of devices for mechanically and automatically cleaning bags of adhering fumes, in the use of instruments for recording and controlling the variables affecting baghouse operation, and in the conditioning of the smoke previous to filtering.

The principles governing bag filtration of fumes were investigated in 1929 by A. J. Seitz of American Smelting and Refining Co., and he arrived at the following conclusions:

1. Pressure on Bags at Constant Volume: The rate at which the pressure builds up in a bag with constant volume is directly proportional to the time, with the same kind of smoke and dust burden (fig. 1).

2. Pressure on Bags for Change in Volume: With equivalent smoke and dust burden, the bag pressure increases with the square of the increase in volume. The rapid increase in pressure is due to (1) increase in volume, (2) increase in amount of dust on the bag, and (3) more intense packing of the dust at higher velocity (fig. 2).

3. Effect of Intensive Filtering on Pressure: Since the pressure builds up directly as the square of the volume, with the same smoke and dust burden, it is necessary to shake the bags in that increased ratio, and vigorously, to keep the pores of the bag open and pressure down. This increases the wear on the bags and the dust losses.

4. Effect of Shaking on Dust Losses: The losses are in direct proportion to the frequency of shaking because the loss takes place immediately after shaking, before the bag seals itself. Tests show that with hand shaking the last 20 pct of the removable dust requires seven times more shaking per unit weight of dust than the first 80 pct.

5. Effect of Fineness of Dust on Losses: The fineness of the dust particle increases the loss in two ways: (1) When the bag is clean a greater percentage of it will pass through before the bag seals itself; (2) fine material penetrates farther into the bag pores, building up pressure faster and necessitating more shaking, thereby increasing the loss.

The physical principle of operation of the bag filter is simply the separation of solids from a gaseous carrier through a porous medium, and is analogous to the filtration of solids from liquid carriers. Basically, the mechanics of the process are as follows: Fume-laden gases drawn from the operating fur-

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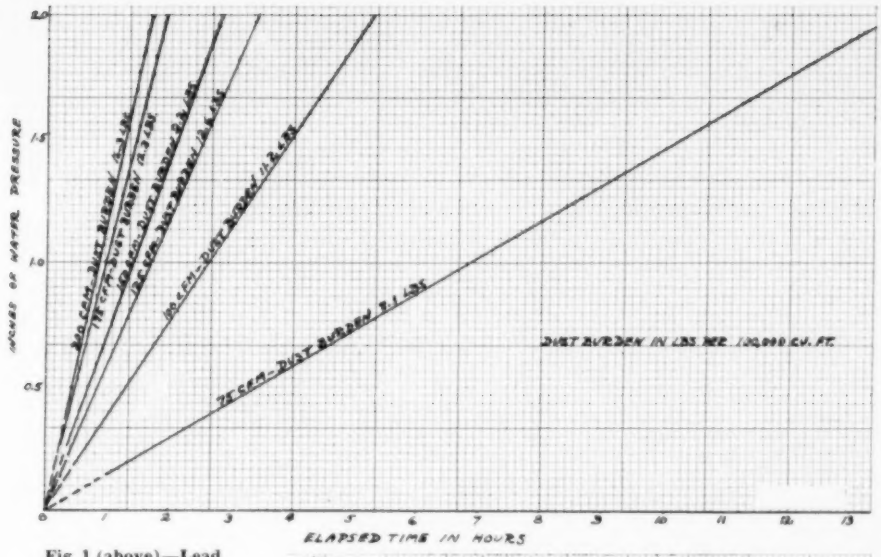


Fig. 1 (above)—Lead D & L smoke pressures for constant volume vs. time elapsed. Standard plain woolen bag 18 in. x 30 ft.

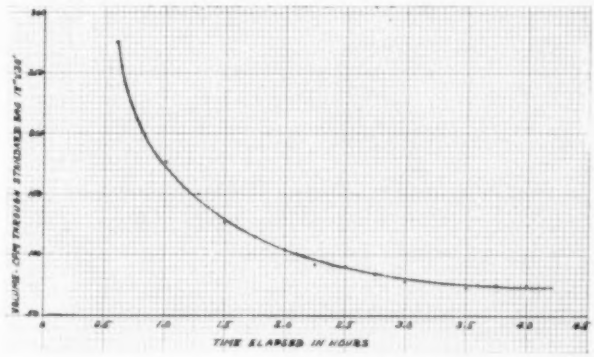


Fig. 2 (right)—Lead D & L smoke.

Dust burden 11 lb per 100,000 cf. Constant pressure of 1.5 in. H_2O. Volume vs. time elapsed. Filtering capacity of plain woolen bag.

nace units by a centrifugal fan, or fans, are blown into the lower chambers of a compartmented baghouse. A sheet steel or concrete floor divides the lower chambers, or cellars, from the upper or bag chambers. Imbedded in the floor are a number of open steel thimbles, usually 18 in. diam and 6 in. high, arranged in a symmetrical pattern. Open bottom ends of tubular cloth bags 30 ft high by 18 in. diam (standard in all A. S. & R. Co. installations for filtration of furnace fumes) are fastened to the outside of the thimbles by wires, straps, or springs and the top end of the bags gathered and tied to steel bells. These bells are suspended by a hook or chain from the horizontal overhead shaker pipes. Waste gases, blown by the fan, ascend into the vertical bags where they are filtered, the dislodged accumulated dust falling to the cellars on shaking and the cleaned gas passing out to the stack. For interrupting gas flow into the chambers while shaking, pneumatically operated dampers are usually employed.

Bag replacement cost is the most important economic factor of baghouse operation, and most efforts toward improving the process have been directed toward increasing bag life. In order of importance, the factors directly involved in length of bag life are (1) temperature of filtered gas, (2) chemical composition of dust and gas, (3) frequency of shaking, and (4) bag pressure.

Experience in the operation of our smelter baghouses has shown that with all other factors equal, woolen bags filtering alkaline, nonsulphurous smoke at a temperature of 150°F, or below, will last almost indefinitely, at temperatures of 200°F, 4 yr, and at 225°F and above, 2 yr. Carbonization of the wool fiber is responsible for rapid deterioration of wool above 150°F. A typical specimen of a bag operated at 200°F will show a brown discoloration and dry and brittle fibers. Under these conditions, the flexing movement of the bag around the thimble during the shaking period subjects the fabric to rapid wear.

The chemical composition of the dust and gas directly influences the life of bag fabric, favorably in the case of alkaline smokes and adversely with acid smokes. Some metallurgical operations, such as lead sulphide sintering, directly evolve SO_3 vapors, possibly through catalytic conversion of SO_2; others, SO_3, chlorides, and selenious vapors. All these acid-forming components of the gas materially attack wool fabric by absorption, particularly in the presence of water vapor.

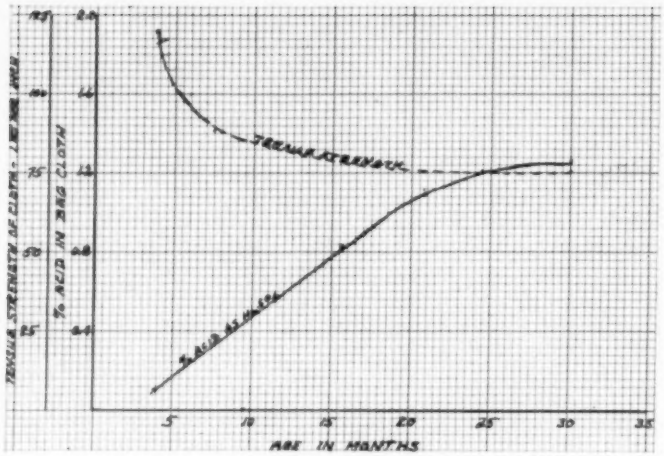


Fig. 3—Woolen fume bags in lead sinter smoke.
Tensile strength and pct acid vs. age in months.

The rate of acid absorption by bag fabric and its effect on tensile strength are shown in attached graph (fig. 3), worked up for a company unit handling lead sinter smoke. Acid absorption by the wool fabric appears to be a straight line function of age up to what may be called the saturation point, but increases more slowly to the ultimate useful life of the bag. Tensile strength decreases in relation to age and acid absorption and becomes practically a straight line near the acid saturation point, when there appears to be no further attack on the already carbonized fibers. A measure of fabric tensile strength alone, however, may give a deceiving picture of the bag's further usefulness, since a fabric showing appreciable vertical resistance to rupture may disintegrate under the flexing motion induced by the shaking mechanism.

Bags are worn mechanically by abrasion of fume sliding down their inner surface during shaking. The effect is cumulative in a downward direction, and the greatest damage is found a foot or two above the thimbles, where the bag fabric is frayed thin and shows greatly enlarged pores. In this region, too, the effect of rubbing against the thimbles while shaking is most harmful. For these reasons, most bag repair work is done close to the thimbles.

To minimize the damage to bags by mechanical wear and reduce the dust loss caused by frequent shaking, the capacity of a baghouse is designed to allow the lowest possible filtering rate per bag consistent with an economic balance between installation cost and bag replacement cost. No rigid figures can be given as to the optimum filtering rate, as they will vary with the dust burden, character, and value of the smoke to be recovered. In our diversified operations we handle dust burdens ranging from 2.5 to 56 lb per 100,000 cf of gas, filtering rates from 50 to 300 cfm per standard bag, and dust varying in density from 15 to 35 lb per cf.

Temperature conditioning of smoke for bag filtration is accomplished by (1) radiation cooling from steel surfaces; (2) cooling by water evaporation; and (3) cooling by admission of atmospheric air. Rarely is one procedure used exclusively, usually a combination of the three is employed.

In most cases, depending on the type of smoke treated, radiation cooling from steel surfaces is not economically effective below temperatures of 450°F, since heat radiation is proportional to the fourth power of absolute temperature difference between inside and outside surfaces of the steel. (Stefan-Boltzman law.) In practice, not only the entering and leaving temperatures across the cooling steel tubes must be considered, but also the burden and tenacity of the fume, for allowance must be made for the periodic rapping of the tubes to dislodge the insulating layer of fume deposited on the inner surface. Under these circumstances the design of a cooling tube installation is largely empirical.

Of the three methods of gas cooling, and within the economical temperature range, radiation cooling of metallurgical smokes by means of inverted, steel U-tubes is to date the most effective, as it introduces no additional volume in the form of water vapor or air to be handled at the baghouse, and, most important, it serves as a permanent buffer to absorb extreme temperature peaks.

A typical example of effective U-tube cooler design is that incorporated into the units used in conjunction with the new El Paso slag treatment

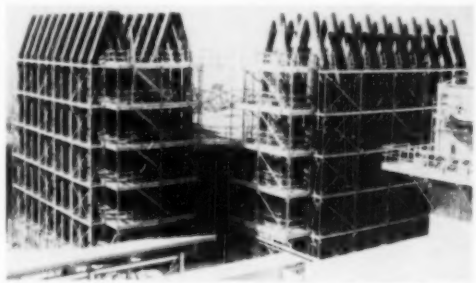


Fig. 4—U-tube cooler. El Paso slag treatment plant.

plant (fig. 4). The tube coolers in this plant consist of two banks in series, each bank containing 20-30 in. x 58 ft U-tubes in parallel-series arrangement. 120,000 cfm of gas enter the first set of tubes at 456°F and leave at 350°F. Entering the second set the temperature is 325°F, some heat having been lost through air infiltration and radiation in the connecting steel flue. Leaving the coolers at 260°F, the temperature is dropped to 225°F entering the baghouse by the admission of air through an automatically controlled damper (fig. 5).

The nature of the zinc oxide fume recovered in this baghouse is such as to preclude the use of cooling by water evaporation, thus the coolers are operated through a temperature range somewhat lower than advocated for economical cooling of other smokes amenable to cooling with water sprays.

Gas cooling by water evaporation is relatively simple, but potentially dangerous when applied to sulphurous smoke. When cooling smoke of this type by water evaporation, we find that a large number of water sprays with low deliveries at 300 psi pressure, or higher, automatically controlled in banks of diminishing number of sprays, is the safest and most efficient arrangement. With relation to the baghouse, the sprays are placed immediately following the radiation coolers, or at the farthest practical distance to minimize the mechanical carry-over of unevaporated water.

The limitations for water-cooling of gas in our baghouse systems are rigid and governed by the resultant dew point of gas entering the baghouse. Generally, the pounds of water vapor per pound of dry gas are calculated so as not to exceed a dew point of 85° to 95°F in the bag chambers.

For final cooling of gas we invariably use atmospheric air in amounts controlled by an automatic damper to maintain the desired bag temperature. Although it requires additional bag capacity to handle the increased volume of air, advantages are that the dew point and dust burden of the smoke are lowered.

Perhaps no greater progress in the bag filtration process has been made than in the mechanics of periodically cleaning bags of accumulated dust. Originally this was accomplished by "airing," that is, allowing the bags to tremble in the current of air created by opening the thimble floor doors and the stack dampers, after shutting off gas flow into the chamber.

Later, came hand shaking of bags. With gas flow shut off, and allowing a few minutes for airing, an operator entered the bag chamber, with or without a gas mask, and gave each bag a vigorous shake. The efficiency of this onerous operation depended too much on the man's strength and willingness and on the supervision.

The next step in the evolution of bag cleaning was the mano-mechanical shaker. Each shaker pipe on which the bags hang was brought through the wall to the outside of the baghouse and, by means of a pivoted lever arrangement, an operator gave each row of bags in a chamber a good shaking by pulling back and forth on the lever.

From this point on, progress in the complete mechanization of the bag shaking process was rapid. It was a single step then to tie all shaker pipes in one chamber to a central shaft outside the baghouse wall and by means of a motor and cam arrangement impart either a vertical drop or a horizontal reciprocating motion to the bags (fig. 6).

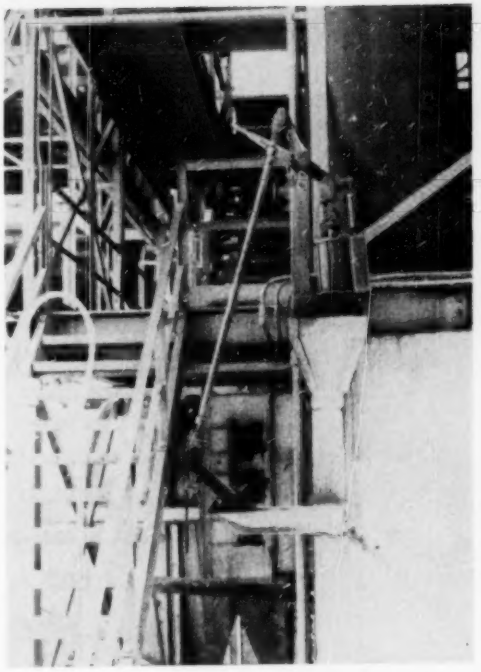


Fig. 5—Automatically controlled damper.

The last step in the modernization of bag shaking was to make the entire process completely automatic. A shaking cycle consists of (1) shutting off gas flow into a chamber, (2) allowing a few seconds for bags to deflate, (3) shutting off damper to stack, (4) shaking of bags, (5) allowing a few seconds for dust to settle in bag chamber, (6) opening damper to stack, and, (7) opening gas inlet to dust chamber. The various steps in the cycle can be automatically initiated in the correct sequence by an electrical contactor consisting of notched cams which actuate solenoid switches to open and close pneumatically operated dampers and to start and stop the shaker motor (fig. 7).

There are four ways in which the electrical control apparatus may be set in motion: (1) by an electric cycle timer set to shake bags at predetermined intervals of time, (2) by a differential pressure contact switch which operates the electrical cam contactor until enough chambers are shaken to reduce the differential pressure across the bags to the predetermined desired value, (3) by a positive pressure contact switch, and (4) by a negative pressure contact switch. Generally, method (1) for actuating the electrical contactor is applicable only to conditions of uniform gas volume and dust burden; the use of methods (2), (3) and (4) is a matter of personal choice, for each one accomplishes the end of maintaining the necessary draft to clear the operating furnace units.

The subsequent treatment and physical character determine the method of removal of dust from the baghouse. Those dusts such as zinc, cadmium, and antimony oxides, which may be considered end-

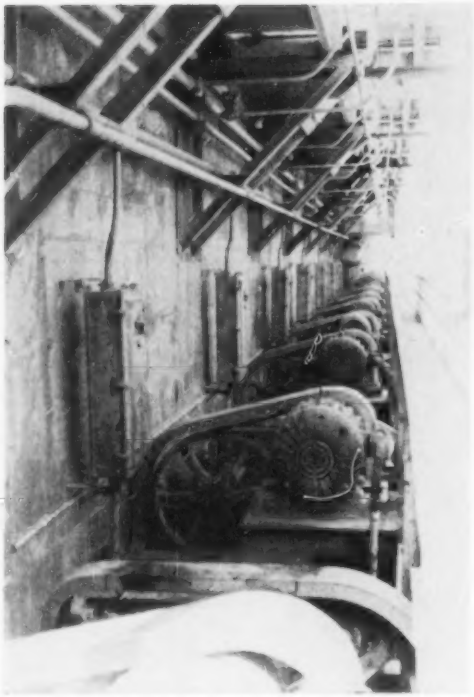


Fig. 6—El Paso slag treatment plant baghouse.

products, and are shipped out of the plant without further treatment, are most economically removed by screw conveyors from steel hoppers under the bag chambers. Pneumatic conveying of this type of dust is sometimes employed, but its use is very limited in our company operations.

Dusts which undergo further metallurgical treatment in the plant, as lead sinter and blast furnace baghouse dusts, must first be burned and clinkered in the baghouse cellars before removing. Clinkering reduces dusting while loading and facilitates subsequent treatment in the smelting units. The densified dust is best removed by mechanical shovels, which method is fast becoming standard in our baghouse operations.

A well organized baghouse operation depends to a large extent on records provided by continuously operating recording and controlling instruments;

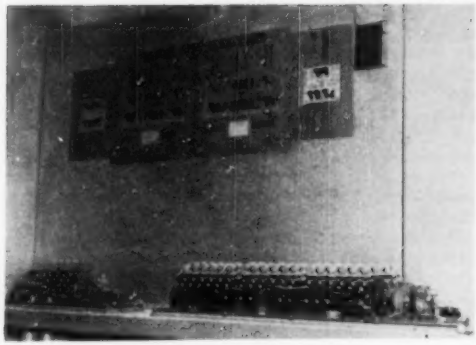


Fig. 7—Electrical contactor.

the use of these devices provides not only a running record of present conditions, but creates a valuable background for future research and design data.

The minimum instrumentation required for operation of a baghouse unit is: (1) bag pressure recorder, which not only shows pressure, but also bag shaking frequency; (2) flue draft recorder, which also shows bag shaking frequency, and (3) bag temperature recorder.

Aside from the variables recorded by instruments, other factors affecting baghouse operation are determined by chemical or physical analysis. These involve the frequent determination of burning rate of dust subject to spontaneous combustion, which must be slowed down by the use of inert materials whenever the tests indicate a dangerously fast burning rate.

In baghouse units handling nonalkaline smokes, daily samples of dust are taken for chemical analysis of relative alkalinity, and the results are used as a guide for the amount of neutralizing agents fed into the smoke stream.

Monthly samples of bags in service are analyzed for acid contents of fabric and tested on standard machines for tensile strength.

In addition to all the precautions in design and operation of a bag filter system, we have developed within our company an automatic apparatus for the continuous determination of dust loss out of baghouse and Cottrell units. The use of this equipment has been an incentive for baghouse men to maintain their operations at the highest possible efficiency.

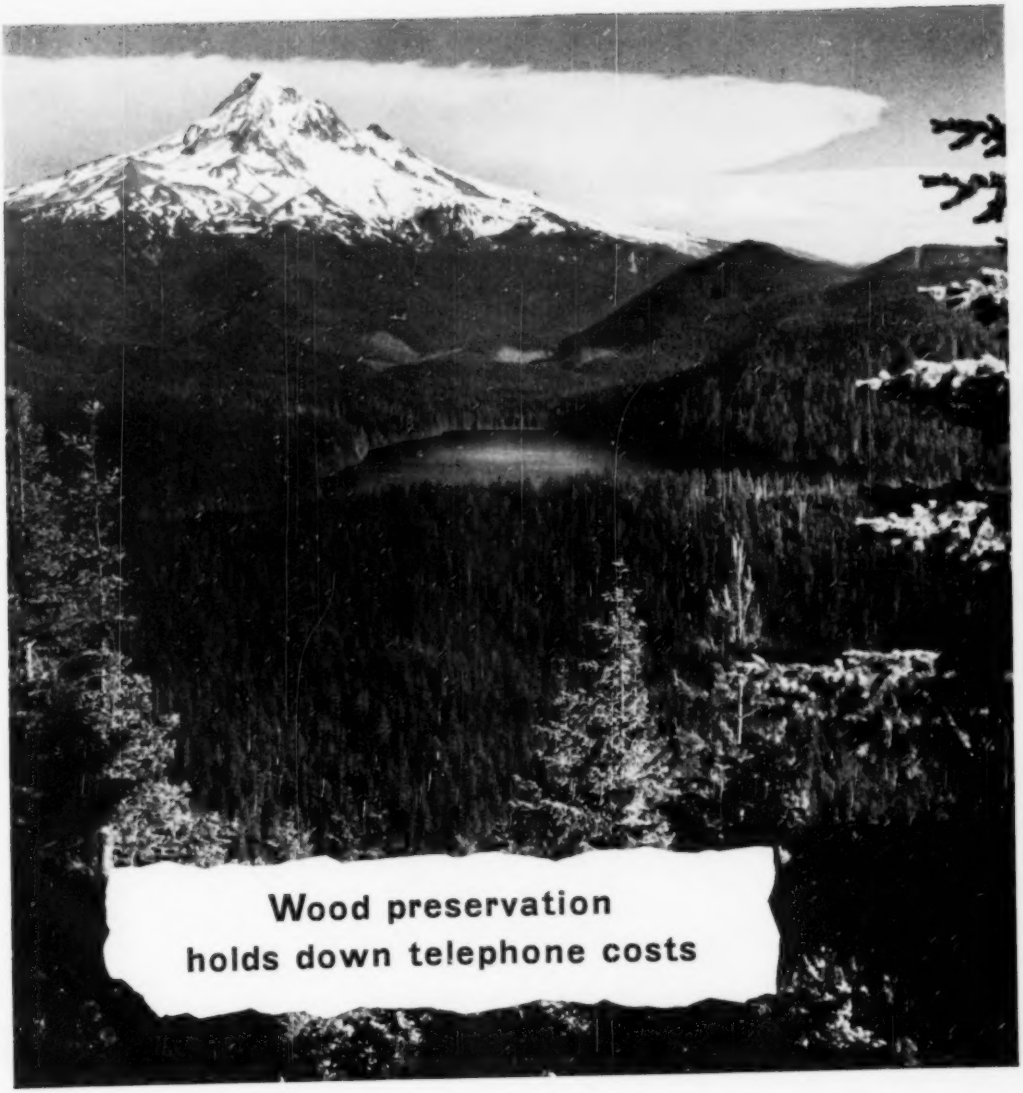
When correctly designed and conscientiously operated, the bag filter process has proved a highly efficient tool in the recovery of metallurgical smokes.

Correction

In Jan. 1950 issue: TP 2739 D. *Fundamental and Practical Factors in Ammonia Leaching of Nickel and Cobalt Ores* by M. H. Caron, p. 78. Following table XI, text should read "will crystallize out as an ammonium compound $(NH_4)_2CO_3 \cdot MgCO_3 \cdot 4aq$. This compound decomposes in boiling water, being converted in $4MgO \cdot 3CO_2 \cdot 4aq$. (Magnesia Alba), NH_3 being released."

Fig. 3, 4, 5 of this paper should appear in TP 2740 D. *Separation of Nickel and Cobalt* and fig. 5 of the latter paper should appear in the paper on ammonia leaching.

TP 2740 D. *Separation of Nickel and Cobalt*. In fig. 1, under 15 and 17, CO should be Co.



Poles are a substantial part of the plant that serves your telephone; making them last longer keeps down repairs and renewals that are part of telephone costs. So Bell Laboratories have long been active in the attack on wood-destroying fungi, the worst enemies of telephone poles.

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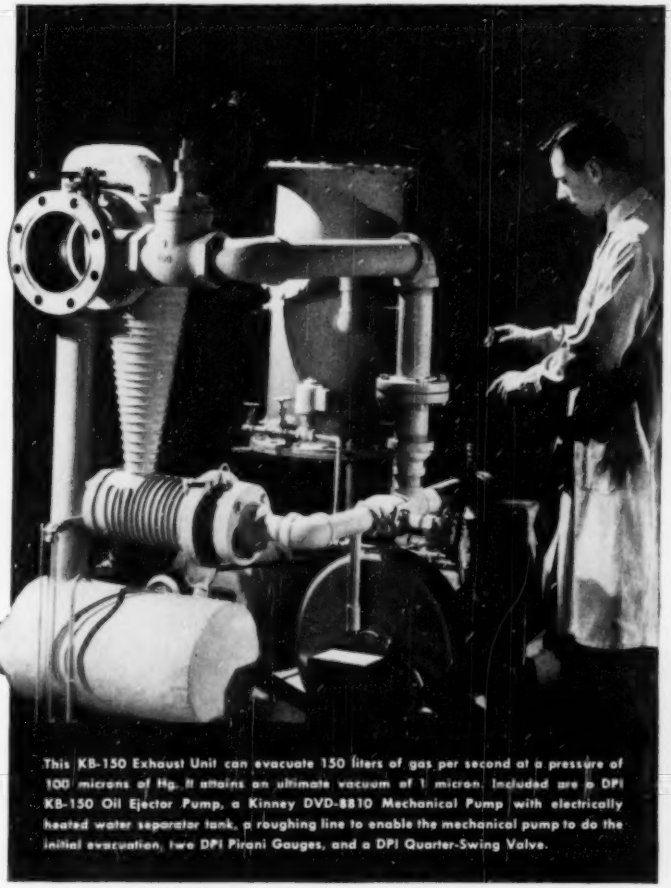

This KB-150 Exhaust Unit can evacuate 150 liters of gas per second at a pressure of 100 microns of Hg. It attains an ultimate vacuum of 1 micron. Included are a DPI KB-150 Oil Ejector Pump, a Kinney DVD-8810 Mechanical Pump with electrically heated water separator tank, a roughing line to enable the mechanical pump to do the initial evacuation, two DPI Pirani Gauges, and a DPI Quarter-Swing Valve.

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